Rockbursts







Case Studies from North American Hard-Rock Mines

Wilson Blake
and
David G.F. Hedley

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

SME

Rockbursts

Case Studies from North American Hard-Rock Mines

Wilson Blake and David G.F. Hedley

Published by the
Society for Mining, Metallurgy, and Exploration, Inc.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

Society for Mining, Metallurgy, and Exploration, Inc. (SME)
8307 Shaffer Parkway
Littleton, Colorado, USA 80127
(303) 973-9550 / (800) 763-3132
www.smenet.org

SME advances the worldwide mining and minerals community through information exchange and professional development. SME is the world's largest association of mining and minerals professionals.

Copyright © 2003 Society for Mining, Metallurgy, and Exploration, Inc.
Electronic edition published in 2009.

All Rights Reserved. Printed in the United States of America.

Information contained in this work has been obtained by SME, Inc., from sources believed to be reliable. However, neither SME nor its authors guarantee the accuracy or completeness of any information published herein, and neither SME nor its authors shall be responsible for any errors, omissions, or damages arising out of use of this information. This work is published with the understanding that SME and its authors are supplying information but are not attempting to render engineering or other professional services.

No part of this publication may be reproduced, stored in a retrieval system, or transmitted in any form or by any means, electronic, mechanical, photocopying, recording, or otherwise, without the prior written permission of the publisher. Any statements or views presented here are those of the authors and are not necessarily those of SME. The mention of trade names for commercial products does not imply the approval or endorsement of SME.

ISBN: 978-0-87335-294-9

Contents

PREFACE v

CHAPTER 1 The Rockburst Phenomenon **1**

CHAPTER 2 Brunswick Mine-Bathurst, New Brunswick **15**

CHAPTER 3 Lake Shore Mine-Kirkland Lake, Ontario **19**

CHAPTER 4 Teck-Hughes and Wright-Hargreaves Mines-Kirkland Lake, Ontario **29**

CHAPTER 5 Lake Shore and Wright-Hargreaves Mines at Depth **35**

CHAPTER 6 Macassa Mine-Kirkland Lake, Ontario **41**

CHAPTER 7 Falconbridge No. 5 Shaft-Sudbury, Ontario **53**

CHAPTER 8 Strathcona Mine-Onaping, Ontario **59**

CHAPTER 9 Creighton Mine-Sudbury, Ontario **65**

CHAPTER 10 Copper Cliff North Mine-Sudbury, Ontario **71**

CHAPTER 11 Quirke Mine-Elliot Lake, Ontario **77**

CHAPTER 12 Campbell Mine-Red Lake, Ontario **83**

CHAPTER 13 Lucky Friday Mine-Mullan, Idaho **89**

CHAPTER 14 Star Mine-Burke, Idaho **97**

CHAPTER 15 Galena Mine-Wallace, Idaho **103**

CHAPTER 16 Sunshine Mine-Kellogg, Idaho **107**

CHAPTER 17 Rockbursting at Kirkland Lake and Coeur d'Alene **113**

INDEX 117

ABOUT THE AUTHORS 121

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

Preface

Mining operations can put extreme stress on rock materials resulting in an explosive and violent reaction, commonly known as a rockburst. Over the past century, rockbursts have become a significant problem in some North American hard-rock mines causing costly mine failures, serious injury, and even death. The sudden, unpredictable nature of rockbursts make them especially challenging to manage. Fortunately, many mines that have experienced rockburst activity have developed effective strategies to deal with this hazard, including controlling blasting times, changing mining methods, altering the sequence of extraction, utilizing backfill, destress blasting, and using yielding supports.

Using a series of case studies, this book attempts to bring greater understanding about the occurrence of rockbursts and the strategies available to prevent them. The introductory chapter defines the types and classes of rockbursts, examines some of the causes of this phenomenon, and provides an overview of how to minimize or alleviate their impact. The book documents the experiences of 15 of the most rockburst-prone mines in the United States and Canada, some of which ceased operations almost 40 years ago. This volume preserves this historical information and experience while providing a beneficial look at state-of the-art solutions to remedy the problem. It is hoped that this text will give mining engineers an indication of when rockburst activity might be anticipated.

The following mining companies are thanked for providing information about the incidents of rockbursts in their mines and allowing us to publish this information:

Kinross Gold for Macassa, Lake Shore, Teck-Hughes, and Wright Hargreaves mines

Noranda Incorporated for the Brunswick Mine

Falconbridge Limited for the Falconbridge No. 5 and Strathcona mines

INCO Limited for the Creighton and Copper Cliff North mines

Rio Algom Limited for the Quirke Mine

Placer Dome Incorporated for the Campbell Mine

Hecla Mining Company for the Lucky Friday and Star mines

Coeur d'Alene Mines for the Galena Mine

Sunshine Mining Company for the Sunshine Mine

CAMIRO (Canadian Mining Industry Research Organization) commissioned this work on rockbursting in North American mines to preserve the information and experience on rockbursts which otherwise might have been lost. Many of the mines have been shut down since the 1960s and a number of mines have been shut down since the 1980s. Special thanks are due Charles B. Graham of the CAMIRO Mining Division.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 1

The Rockburst Phenomenon

INTRODUCTION

Rockbursts in hard-rock mines are a twentieth century phenomenon. They first occurred at the turn of the century in the gold mines on the Witwatersrand in South Africa and on the Kolar Gold Field in India. In North America the first occurrences were in the early 1900s in the lead-zinc-silver mines of the Coeur d'Alene Mining District of Idaho and the copper mines on the Upper Peninsula of Michigan. In Ontario the gold mines at Kirkland and the nickel mines in the Sudbury Basin experienced bursting in the 1930s, followed by the gold mines at Red Lake in the 1960s and the uranium mines at Elliot Lake and the lead-zinc mines in New Brunswick in the 1980s.

Rockburst case histories have been compiled for 15 mines in Canada and the United States. These include the Brunswick lead-zinc mine at Bathurst; Lake Shore, Teck-Hughes, Wright-Hargreaves, and Macassa gold mines at Kirkland Lake; Falconbridge No. 5, Strathcona, Creighton and Copper Cliff North nickel mines in Sudbury; Quirke uranium mine at Elliot Lake; Campbell gold mine at Red Lake; and Lucky Friday, Star, Galena, and Sunshine silver-lead-zinc mines in the Coeur d'Alene. In general, these are the mines that experienced the most significant rockburst activity. In addition, these mines have well-documented records of rockburst occurrences and published reports that analyze the probable cause and mechanism.

Some other hard-rock mines in North America have had similar rockburst activity, but documentation is lacking. The most notable of these include the Director fluorspar mine in Newfoundland, which experienced rockbursts at a depth of only 150 meters (m), and the granite quarries of Vermont, which had large bursts right on surface. East Malartic gold mine in the Val d'Or area of Quebec had many bursts, up to 4.1 M_n. Kerr Addison gold mine near Kirkland Lake was forced to change from shrinkage to cut-and-fill mining to better control their rockburst problem. During the 1930s and 1940s Frood-Stobie Mine in Sudbury was as active as Lake Shore Mine in Kirkland Lake. Little Long Lac gold mine in northwestern Ontario had a significant rockburst problem, with total production of only 1.6 million t. Homestake gold mine in South Dakota started to experience rockbursts on the lower levels, at depths of greater than 2,100 m, after almost 100 years of mining. White Pine copper mine in Michigan, and Climax molybdenum mine in Colorado both experienced a few large bursts, greater than 3.0 M_l, apparently as a result of large collapses.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

DEFINITION OF A ROCKBURST

Definitions to characterize the occurrence of a rockburst have been made by a number of governmental agencies. The following definitions do not identify the conditions or causes responsible for bursting.

- "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" (U.S. Bureau of Mines [USBM] 1968).
- "An instantaneous failure of rock causing an expulsion of material at the surface of an opening or a seismic disturbance to a surface or underground mine" (Ontario Ministry of Labour 1983).
- "A sudden and violent failure of a large volume of overstressed rock, resulting in the instantaneous release of large amounts of accumulated energy" (Mine Safety and Health Administration 1984).

These definitions include everything from the popping off from the face of a few small pieces of rock to seismic events that displace large volumes of rock underground and are felt on surface, similar to a local earthquake.

Rockbursts that occur in soft-rock mines—coal, potash, and trona—are referred to as "bumps." In fact, the worst rockburst disaster in Canada occurred at the Springhill coal mine in Nova Scotia in 1958, resulting in 75 fatalities. The largest rockburst in North America, 5.2 M_l, occurred at the Solvay trona mine in Wyoming in 1995, when an entire panel, 1,000 m by 2,000 m, collapsed. Coal mines in the United States have experienced bumps since the 1920s, with magnitudes up to 4.5. In the last 60 years, there have been 78 fatalities in coal mines due to bumps, compared to 22 rockburst fatalities in the hard-rock mines of Coeur d'Alene. However, rockburst occurrences in soft-rock mines are beyond the scope of this work.

ROCKBURST CLASSIFICATION AND TYPES

Soon after the start of rockbursting in Ontario, a classification system was developed and applied provincewide to record and report these bursts. The main criterion was the amount of damage, but the amplitude and duration of the felt vibrations were also taken into account. The "light" category was less than 10 t of displaced rock, "medium" 10–50 t, and "heavy" greater than 50 t. Any bursts involving fatalities or injuries were also reported, irrespective of the amount of damage. In the Coeur d'Alene, following an American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc. (AIME) sponsored symposium on rockbursts in 1941, the terms *induced bursts* and *inherent bursts* were adapted from Canadian terminology. There was no local or national agency that kept track of rockbursts; however, most mines maintained their own burst records.

In the early 1980s the Ontario system was augmented to include *magnitude*, similar to earthquake classification. Magnitude values are obtained from seismographs installed by the Geological Survey of both Canada and the United States. In western North America a local Richter earthquake scale (M_l) is used, and in the eastern part of Canada a Nuttli earthquake scale (M_n) is used. Both scales are logarithmic and agree to within ±0.3.

Large Seismic Events

Underground and close to the source, rockbursts are heard as loud, distinct blast-like noises. From a farther distance, they sound like deep, dull thud-like noises. When a large

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

volume of rock has been displaced by a burst, there is generally a shock wave of air that is felt in nearby interconnected openings. Therefore, most underground miners have always referred to rockbursts as "airblasts." On the surface, rockbursts sound like a sonic boom, but both the ground and surface structures shake similarly to what would be felt from a small, nearby earthquake.

Underground damage from rockbursts is basically from two different mechanisms: (1) the effect of the burst itself and (2) the effect of the resulting seismic wave. Rockbursts occurring in overstressed structures, particularly pillars, result in the breaking up of the volume of rock involved into small fragments, create a very dusty atmosphere, and do various degrees of damage to the walls and supports in adjacent openings. Damage is generally very intense over a relatively small area, as shown in Figure 1.1.

The seismic wave created causes the slabbing off of large pieces of rock from the walls and back of nearby openings, as well as the shake-down of loose rock. Ground support in openings, particularly timber or steel, can be severely damaged by the effects of the passing seismic wave. Typical seismic wave damage is shown in Figure 1.2. This type of damage can be limited or very widespread depending on the magnitude of the burst.

The seismic waves generated by a rockburst can be detected on surface seismographs over great distances. Small- to medium-sized bursts are usually detected within a 100-kilometer (km) radius, while large bursts have been detected at distances greater than 1,000 km. The seismographic trace of a nearby rockburst is shown in Figure 1.3.

Small Seismic Events

Popping, spitting, and sometimes spalling are stress-induced surface rock failures that also generate seismic waves. The volume of rock involved in such small-scale failures is usually less than 0.5 cubic meters (cu m), and no airblast or shock wave is created. The seismic and audible signals generally cannot be heard or detected much beyond 50 m from the source.

Bumps or *knocks* are terms applied to noises heard in the mine but not generally heard or felt on surface. If they occur close to an opening, they can displace a volume of rock generally in the 1- to 2-cu-m range.

Types of Rockbursts

The mines also categorized rockbursts according to their underlying causes. From a theoretical standpoint, a differentiation was made between volume and contact failures. Volume failure was also subdivided into those that were localized and the collapse of complete pillars. In mining technology, these types were labeled *strain*, *pillar*, and *fault-slip* bursts.

Strain bursts are caused by local high-stress concentrations at the edge of mine openings. They are normally associated with the driving of shafts, development headings, and stope faces. Generally, the magnitude of these bursts is less than 2.0, and damage is less than 100 t. They are common to all rockburst-prone mines and frequently occur during drilling for blasting or reinforcement. The rock ahead of the advancing face or heading is unable to adjust to an immediate stress increase as a result of the blast, and therefore, these small-scale failures occur. They generally occur along a geologic structure, such as a joint, and the resulting damage is usually controlled or limited by an obvious structure.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.



Sill drift



Timbered raise area

FIGURE 1.1 Examples of damage from sill pillar bursts in cut-and-fill mining

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.



FIGURE 1.2 Shock-wave damage to timbered section of a drift

Pillar bursts occur when the stress exerted on a pillar exceeds its strength. When the pillar fails, a large amount of stored strain energy is released. In addition, the failure of the pillar results in instantaneous closure of the walls or roof and floor that releases an even larger amount of potential energy. The actual point of failure may be in the immediate walls of the pillar and occur on a geologic structure, the weakest link in the high-stress area. In mining, there are a number of different types of pillars. Rockbursts in shaft pillars have occurred in all four gold mines at Kirkland Lake. Bursts in sill pillars of shrinkage and cut-and-fill stopes were prominent at all four mines in the Coeur d'Alene and at the Kirkland Lake, Campbell, and Creighton mines in Canada. Examples of bursts in regional sill pillars include the Strathcona and Brunswick mines. Mines using variations of room-and-pillar methods can be subject to a chain reaction of pillar failures accompanied by rockbursts. Two such occurrences have taken place at the Quirke Mine at Elliot Lake and the Campbell Mine at Red Lake. Generally, pillar bursts can have a magnitude up to 3.5, and the damage involved can be hundreds of tonnes. Pillar bursting is most common when the extraction is greater than 80%. That is, a small pillar is left supporting a large mined-out area.

Fault-slip bursts occur when the shear stress along a geological structure exceeds the normal stress or clamping force acting on the structure. These types of bursts have occurred at Lake Shore and Wright-Hargreaves mines at Kirkland Lake; Falconbridge No. 5, Strathcona, and Creighton mines in Sudbury; and Lucky Friday and Star mines in the Coeur d'Alene. They mostly occur when the mined-out area of a stope or mine becomes very extensive, greater than 1 sq km. A subset of fault-slip bursts is a pillar foundation failure, where shearing takes place through intact rock rather than along an existing weakness plane. Only a few instances of these types of bursts have been reported in North American mines, all occurring at the Macassa Mine beginning in 1996.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

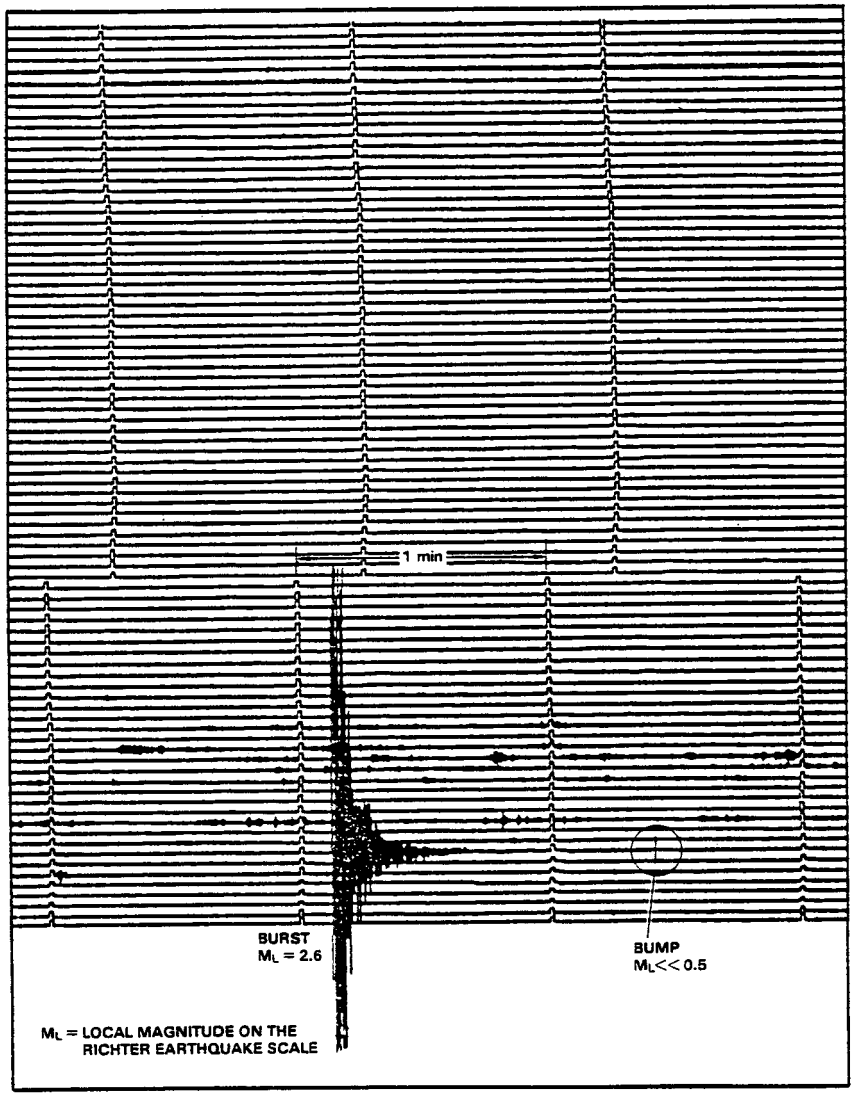


FIGURE 1.3 Seismographic record of nearby rockburst

Magnitude of fault-slip bursts can be very large. At Lake Shore a 4.4 M_n rockburst caused damage over multiple levels involving thousands of tonnes of displaced rock. However, at the Lucky Friday a 4.1 M_l burst displaced only a few hundred tonnes in the adjacent stope being mined and a similar amount of shake-down damage in an old mined-out stope some 150 m above. The very large fault-slip rockbursts are usually related to the overall mining geometry and stress transfer or closure effects and not to the day-to-day mining cycle. For this reason, these large bursts are not generally triggered by mine blasting and, hence, can occur at any time.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CAUSES OF ROCKBURSTING

As an excavation progresses, the stresses change and the rock deforms around the mine opening. For any type of seismic phenomena to occur, including rockbursting, the stress changes caused by the excavation must result in an unstable equilibrium of the rock mass.

Violent Failure Versus Controlled Failure

When the stress in a rock structure exceeds its strength, the structure may fail. Whether it fails suddenly and violently or gradually and nonviolently depends on the physical properties of the rock as well as the characteristics of how it is being loaded or stressed.

For the case of a mine pillar, whether it fails violently or nonviolently depends on the postfailure stiffness of the pillar compared to the stiffness of the loading system (mine stiffness). If the loading system is softer than the pillar stiffness, the pillar will fail violently once its strength is exceeded. If the pillar stiffness is softer than the loading system stiffness, the pillar will deform and fail gradually as its strength is exceeded. The stiffness of the loading system, mine stiffness, is controlled by the span between pillars, the size and location of pillars, and the deformation modulus of the wall or roof rock. As the span between pillars increases, the mine stiffness decreases.

Geologic Settings Favorable for Rockbursting

Most of the hard-rock mining districts throughout the world that experience rockbursting have many similarities. They are in an initially high-stress environment owing to depth, tectonic forces, or a combination of both. The shape of the orebody is tabular, and the thickness of the vein or width of the seam is usually less than 5 m. The direction of the major principal stress is usually perpendicular to the tabular dimension. The extraction ratio is high, usually greater than 80%. The wall rocks, and often the vein or seam material, are very hard, brittle, and strong, as well as being very old geologically—mostly of Precambrian age (greater than a billion years old). In addition, the geology is usually very complex with respect to folding, faulting, and metamorphism. This combination of high stresses, narrow veins, high extraction ratios, and strong brittle rocks in a complex geologic environment is almost always certain to lead to the occurrence of rockbursting in deep mines.

Quartzites and metamorphosed intrusive and extrusive rocks are most burst prone followed by igneous intrusives. Quartzites are the wall rocks on the Witwatersrand in South Africa, the Coeur d'Alene in Northern Idaho, and at Elliot Lake in Ontario. Basic syenites, syenite porphyry, and tuff are the wall rocks at Kirkland Lake; norites, granites, and greenstones are the wall rocks at Sudbury; and andesites, basalts, and rhyolites are the wall rocks at Red Lake.

DEALING WITH THE OCCURRENCE OF ROCKBURSTING

Though there is no practical way to eliminate rockbursting, measures have been developed that have been found to minimize both the occurrence and effects of bursting.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

Geometric Considerations

The most common means of minimizing high stresses around mining openings is by using geometries and excavation sequences that do not create burst-prone pillars and that transfer stress to abutments in a manner that reduces stress concentrations. In practice, this is achieved by utilizing a longwall-type mining geometry. Any pillars left must be large enough so that they are not burst prone. The South Africans converted to underhand longwall mining geometries in the 1950s, and the Lucky Friday was the first North American mine to convert to an underhand longwall in the 1980s. Burst-prone sill pillars are recovered primarily by destressing in the Coeur d'Alene, whereas Canadian mines tend to leave larger sill pillars and recover them by long-hole techniques. In many mines burst-prone sill pillars are left unmined in order to prevent uncontrolled bursting. Utilizing numerical modeling techniques, mining geometries and sequences can be determined that minimize both the creation of high stresses and the energy release caused by mining.

Geologic Considerations

As mining experience in an orebody or district is gained, relationships between rockbursting and certain geologic features become evident. Dykes, faults, specific stratigraphic horizons, or a geologic unit are examples of geologic features that result in high-stress concentrations and are very burst prone when intersected by either mining or development openings. Mining through or along such features should be carried out so that stress increases are minimized or dealt with by destressing.

Extraction Rate

As a face is advanced in highly stressed ground, a fracture zone is developed ahead of this face. While most of this fracturing occurs within a few hours after the face is blasted, the fracture boundary zone continues to migrate for some 24 hours. The depth of this fracture zone depends on the face height or width and on the abutment stress. This fracture zone is a self-destressed zone between the opening and solid intact rock.

Too rapid a face advance does not allow time for this fracture zone to occur gradually and reach stress equilibrium. If stress equilibrium cannot be reached, the occurrence of a rockburst is likely. In many mines, stope faces prone to bursting are advanced on a single-shift basis, or are temporarily halted until the ground has stopped working and has settled down. Destressing is often used to prevent ground working and/or work stoppages.

In addition, the length of round may be reduced to minimize the volume of rock affected by the stress transfer from blasting. The South Africans went to a very short face advance, incremental-type mining, to deal with their rockburst problems. Most mines in North America converted to cut-and-fill-type mining to minimize the geometry change caused by the daily face advance. More recently, mines have converted to long-hole-type mining to reduce exposure at the face and increase productivity. In many cases, this conversion has changed the character of the bursting and resulted in larger bursts. This happened at Macassa when the mine reopened in 1994 following a double fatality.

Reinforcement to Contain the Effect of Bursting

Traditionally reinforced backs and walls of openings will not hold up under the effects of a nearby rockburst. Such reinforcement is designed to support dead loads and not the dynamic shock loads resulting from a rockburst. Rigid support or rock reinforcement is

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

not able to yield and, hence, fails or deforms almost instantaneously during a burst, allowing the back or walls to come in and fill the opening.

Yielding support that can absorb energy during deformation has been found to best contain rockburst damage, particularly dynamic shock damage. Such support systems are designed to hold the rock together within the fractured zone around an opening, and to deform along with the fractured rock. Containment of damage maintains the integrity of the access and prevents miners from being buried. The Lucky Friday Mine has achieved great success with a yieldable support system consisting of split sets and resin-grouted Dywidag bolts, along with chain link mesh (see Figure 1.4). This reinforcement is supplemented by a form of cable lacing for additional containment support at the intersection of a stope and an access opening. In "bumping" ground, steel-fiber–reinforced shotcrete is applied to the walls and back prior to installation of bolts and chain link mesh.

Destressing

In mining, it is not possible to prevent the occurrence of most rockbursts. For some special geometric relationships, however, it has been possible to avoid the occurrence or trigger a burst by destressing. The idea of destressing is to blast fracture the rock in a high-stress or potential rockburst zone to reduce the stress. Destress blasting reduces the stress and dissipates stored strain energy by inducing fractures that allow yielding.

The most successful application of destressing has been with respect to development headings such as tunnels, drifts, and shafts. The usual procedure is to drill and blast at least two parallel holes, drilled ahead of the face, and two holes angled out into the walls.

This prevents the high stress ahead of the face from forming after a blast and thereby eliminates or controls the popping, bumping, and small bursts that might occur. This practice is routinely used in the deep mines in the Sudbury region when development openings are being driven in granite or other brittle rock types.

Destressing of burst-prone sill pillars is also carried out. The pillar is usually line drilled and the holes blasted to fracture it and allow it to yield. This practice has been very successfully used in the Coeur d'Alene. The timing and sequencing of destressing must be well planned. Stress transfer from destressing has caused bursts in adjacent highly stressed mine structures, and waiting too long to destress has resulted in bursts triggered by the destress drilling. The last rockburst fatality at the Lucky Friday, in 1986, was triggered by destress drilling.

Rockburst Monitoring

Most mines experiencing rockbursting have installed some type of seismic and/or microseismic monitoring system. Data from seismic monitoring are used to assess the relative seismicity and potential rockburst hazard associated with advancing mining faces. Knowledge of a particular problem area and its causes allows rockburst control measures to be taken to minimize both the occurrence and effects of a burst. Data from seismic monitoring is also used to determine reentry time following long-hole blasts, and the rapid location of a rockburst and damage assessment are vital in initiating rescue operations.

Rockburst prediction was the goal of microseismic monitoring when the technique was first developed by the USBM in the late 1930s. Although this monitoring technique has been successful in detecting and delineating burst-prone mine structures, use of this

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.



Crosscut access above stope



Burst-induced closure within stope

FIGURE 1.4 Contained damage with yielding ground support

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

data to determine if a structure is going to burst and when it will burst has not been possible except in extremely rare instances.

SOME GENERAL RULES FOR ROCKBURST OCCURRENCES

With a 100-year database, it is possible to formulate some general rules concerning rockburst occurrences. Some rules are universal, while others are mine or mining district specific. As always, there are exceptions. Four main parameters control rockburst occurrences: rock properties, depth, aerial extent of mining, and rate of production:

1. A strong, brittle, elastic rock is a requirement, which is typical of the host rocks for all 15 mines described in this report. It is not so much the energy these rocks can store, but rather their postfailure characteristic of a very steep drop in stress with very little displacement. In some of the Sudbury mines, the massive sulfide orebodies are relatively weak and soft and crush in a nonviolent manner. In these cases, rockbursts are generally confined to the stronger disseminated ore zones, waste inclusions, or wall rocks.
2. Depth controls the magnitude of the in situ stresses. In the Canadian Shield, as well as the Coeur d'Alene, the major and intermediate principal stresses are oriented in a subhorizontal direction, with the vertical gravitational stress normally being the minor principal stress. Typically, the major principal stress can be up to double the vertical stress.
3. Aerial extent of mining controls where stress is transferred and its concentration in pillars and abutments. It probably also controls the maximum size of rockburst that can occur, especially for fault-slip bursts, because it determines the volume of rock that can be involved.
4. Rate of production controls the rate of change in stress on rock structures. It appears to be a very important parameter, because when mines close, rockburst activity ceases immediately or declines rapidly, although the other parameters remain constant. Most mines that convert from cut-and-fill- to long-hole-type mining experience larger rockbursts. The change from small face advances to large face advances results in more widespread stress transfer effects. This often results in the occurrence of large bursts out in the walls on structures rather than small bursts ahead of the face. In some mines, notably those at Kirkland Lake, rate of production seemed to be the predominant factor controlling rockbursts below a certain depth. At the Lake Shore Mine, from a depth of 1,200 m to almost 2,500 m, damaging rockburst activity was relatively constant at 1 per 7,400 t mined. At the Lucky Friday Mine, the cumulative seismic energy released per cumulative tonnes mined, on an annual basis, has been almost a straight line since 1988. Mining during this period progressed from the 5100 level down to below the 5900 level and has been in the same wall rock. Rate of production has no effect on rockbursts in old abandoned areas of mines, such as the chain reaction of pillar failures at the Quirke and Campbell mines. In those cases, past mining practice was responsible.

A universal rule is that small rockbursts are more numerous than large bursts. For Ontario mines, between 1931 and 1971, 3,732 bursts were reported, of which 11% were classified as "heavy," 16% as "medium," and 73% as "light." A similar relationship exists with respect to magnitude. Between 1984 and 1994, 488 bursts were recorded, of which

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

6% were of 3.0 M_n and over, 32% of 2.5 to 2.9 M_n, and 62% of 2.0 to 2.4 M_n. Though districtwide rockburst statistics are not kept for the Coeur d'Alene, similar percentages for the distribution of burst magnitudes are found at each mine. With the advent of microseismic monitoring systems, many thousands of very small seismic events are recorded, of which perhaps less than 1% are damaging rockbursts.

For some mines with a fairly large rockburst database, a logarithmic relationship between magnitude and frequency of occurrence, over a specific time period, can be established. From this information, the average recurrence time for various magnitudes can be calculated, to produce a type of seismic hazard assessment. For instance, over an 8-year period at the Creighton Mine, on average a rockburst of at least 2.0 M_n occurred about once a month and a rockburst of at least 3.0 M_n occurred about once a year. There are limitations to this type of analysis. A basic assumption is that the driving force (i.e., the four main parameters) remains relatively constant. At Quirke Mine, when the hanging wall fractured though to surface, the stress regime was radically altered, and the distribution and frequency of rockbursts before and after were completely different. Similarly, at Lucky Friday when the mining front entered the Lower Revett Formation, the incidents of rockbursts increased by a factor of about 10, owing to a change in the character of the wall rock.

Another limitation concerns the characteristics of the very large rockbursts, those of 3.5 M_n and greater. There are numerous examples where these large bursts do not occur in isolation, but rather generate multiple events. At Lake Shore, on six occasions, large rockbursts of almost equal magnitude occurred minutes apart. Other examples include Wright-Hargreaves in 1964, Campbell in 1983, and Falconbridge No. 5, Creighton, and Quirke mines in 1984. More recently, at the Macassa Mine in April 1997, a 3.8 M_n burst was followed by a 3.7 M_n burst about 3 hours later, and at Creighton in May 1998 bursts of 3.9 M_n and 3.5 M_n occurred about 12 hours apart. Similar occurrences of multiple large bursts have not been observed in the Coeur d'Alene. At Macassa it took 13 days for the mine to return to the normal background microseismic rate; whereas, at Creighton this was achieved in 2 days. These characteristics, multiple large events and the time required to return to normal background rate, will affect the reentry policy into burst areas at each individual mine.

Both magnitude and damage are related to the peak particle velocity of the resulting seismic wave. It could be anticipated that there would be a relationship between burst magnitude and the amount of damage. On a tonnage basis, it is not so clear cut, although generally the amount of damage increases with magnitude, but there are numerous exceptions. Rockbursts that occur out in the wall rocks, at some distance from mine workings, even of large magnitude, generally only result in shake-down of loose rock. It would appear that the size of the mine openings, the type of ground support, and the proximity to the burst control the amount of damage. In some mining camps there is a clear relationship between burst magnitude and the extent of damage. For Kirkland Lake, a 3.0 M_n rockburst would be expected to produce damage over a vertical distance of about 170 m, which would double for a 4.0 M_n rockburst. Here the vertical distances are used, because, in most cases, levels were blocked off and lateral extent of damage could not be established. However, if a circular shape is assumed, it means that the aerial extent of damage quadruples between 3.0 and 4.0 magnitude events. This type of relationship may also be applicable to other mining camps.

Accurate prediction of rockbursts is not feasible. However, at some mines up to 80% of bursts occur with or shortly after blasting. At these mines, central blasting at the end of

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

a shift is employed. In some cases, there is warning of impending rockburst activity. The clearest example is before a chain reaction of pillar failures, when pillar deterioration precedes rockburst activity by a matter of months. Where bursts are caused by high-stress concentrations, there are numerous examples where the work force has been withdrawn, due to working ground, prior to a rockburst. As far as is known, there has never been any warning for fault-slip rockbursts. This would imply that the triggering mechanism is a reduction in the clamping force rather than an increase in the shear stress.

Old Rules for Rockbursts That Still Apply

A number of rules to control rockbursting have been listed by different organizations since the 1920s. Many of these rules still apply and are often forgotten. Hence, a listing of some of these old rules is repeated here.

- A properly planned sequence of stoping for the whole orebody should be adopted and followed as closely as possible.
- The merging of large excavations at depth should be avoided.
- Pillars should be eliminated or reduced to a minimum.
- Parallel veins should be stoped singly, the hanging wall vein first (footwall vein first if underhand mining).
- Where veins branch, stoping should begin at the intersection and mine away from it, one branch at a time.
- Where possible, stoping should proceed away from a fault or other plane of weakness.
- Slowing the rate of extraction will often reduce the amount of bursting in relation to tonnage mined and may actually prevent bursting under some conditions.
- Mined-out areas should be filled, and filling should proceed concurrently with extraction and be kept as close to the face as possible.
- Shafts should be sited to avoid intersecting the orebody.

REFERENCES

Mine Safety and Health Administration. 1984. Hearing on Proposed Standards for Ground Control Metal/Nonmetal Mines, Spokane, WA, June 19, 1984.

Ontario Ministry of Labour. 1983. *Occupational Health and Safety Act and Regulations for Mines and Mining Plants*. Reg. 694, No. 34a. pp. 48.

U.S. Bureau of Mines. 1968. *A Dictionary of Mining, Mineral, and Related Terms.* Washington, DC: U.S. Bureau of Mines. pp. 932.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 2

Brunswick Mine Bathurst, New Brunswick

SUMMARY

The Brunswick Mine started to experience ground control and rockburst problems in the mid-1980s when sill pillars were reduced below some 40 m. To deal with this problem, overhand cut-and-fill mining was switched to long-hole panel mining to recover sill pillars. With increased depth and increasing seismicity, the mine converted to a primary-secondary panel mining geometry. Continued ground problems associated with the panel and pillar geometry in the mid-1990s resulted in significant ore losses as well as an increased hazard. To alleviate this problem, the mining method has again been changed to a pyramidal pillarless mining front where possible.

GEOLOGY AND MINING METHODS

The massive sulfide orebody at the Brunswick Mine consists of up to 10 subparallel lenses striking north-south and dipping some 75° to the west. The overall strike length of the orebody is about 1,200 m, with a width of up to 200 m, extending from surface to a depth of about 1,200 m. The orebody is located in a metamorphosed volcanic sequence of sediments and tuffs. The chloritic and sericitic footwall metasediments are more competent than the hanging wall sequence of chloritic sedimentary units and chloritized tuffs. The metasediments, particularly in the hanging wall, are very schistose with phyllitic partings. Hence, the hanging wall and footwall rocks are much softer and weaker than the massive sulfide ore, which has an unconfined compressive strength of about 210 megapascals (Mpa). An intrusive porphyry dyke runs along the orebody, mainly in the footwall, but it does cut across the south end of the orebody.

The major principal in situ stress is subhorizontal and is oriented east-west, perpendicular to the strike of the orebody. The intermediate principal stress is also subvertical but is oriented north-south. The major principal stress is 1.9 times the overburden stress, and the intermediate principal stress is 1.6 times overburden stress.

Above the 475 level, the orebody was mined by primary-secondary open stoping. The mining method was changed to mechanized cut-and-fill in the early 1970s, and overhand cut-and-fill mining was initiated from the 575, 725, and 850 levels. As the mining from the 575 level progressed upwards and the first sill pillar was formed, mining conditions became more difficult in the more highly stressed ground. This brought

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

about a change back to primary-secondary open stoping by the mid-1980s. Mining on the 1000 level was also started with primary-secondary open stoping. Open stopes were filled with rock fill rather than hydraulic sand fill. Initially, primary mining of large open stopes went well, but as the extraction ratio increased, both stress and ground control problems increased. By mid-1996, it became difficult to maintain the 9,500 t per day production, owing to ground instability problems. Critical accesses to crucial production areas were being damaged beyond repair, scheduled stopes were caving in, and high seismicity was preventing access to active production horizons. Hence, the mining method was changed to a pyramidal pillarless mining front where possible. Further, chronic difficulties and shortages were encountered in filling large open stopes with rock fill. Paste backfill was introduced to replace rock fill, as well as to allow tight filling of difficult-to-access caved areas.

ROCKBURST INCIDENTS

In the early 1980s seismic events of greater than 2.0 M_n began to be induced by mining as sill pillars were reduced below 40 m. As a result of a rockburst sequence in August 1984 and a 3.5 M_n event a few months later, the mine decided to install a microseismic monitoring system to investigate the rockburst problem. In early 1986 a 32-geophone system was commissioned to monitor the sill pillars above the 725 and 850 levels.

Owing to increasing stress and burst occurrences, it became obvious that overhand cut-and-fill mining in the wide stopes above the 725 level could not be safely carried out to reach a designed 15 m sill pillar. Therefore, the mining was changed back to a primary-secondary open stoping above the 725, as well as the 850 and 1000 levels.

The seismicity mostly occurs within the hard and brittle massive sulfide horizons. Many different source mechanisms appear to be associated with the induced seismicity and rockbursting. The most apparent are high-stress concentrations in pillars, large contrasts in stiffness between adjacent rock types, and slippage along discrete structural features. Figure 2.1 shows a long section of Brunswick and the locations of seismicity associated with mining during the year 2000.

It is apparent that the seismicity and rockburst problems are primarily associated with the formation of pillars. Large sill pillars result from a multilevel mining front, and small stope pillars result from the primary-secondary stope geometry. Rockbursts associated with stoping can result in massive back failures in the large open stopes, as well as damage to secondary pillars. Strain bursting can occur along sill level extraction drifts in the footwall where small pillars are formed by intersecting openings. Wall bursting in the massive sulfides can also occur when sudden slips take place along bands of weak sedimentary layers present within the ore zone.

DISCUSSION

The mining methods used over the life of the mine have been mechanized cut-and-fill, primary-secondary open stoping with delayed backfilling, end slicing (modified avoca), and more recently, pyramidal pillarless open stoping with paste backfill. As the extraction ratio increased, the operation became more and more susceptible to stress-related problems, including rockbursts, which have caused large tonnages of reserves to be written off or put into a delayed mining category. To deal with these problems, the mine has

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

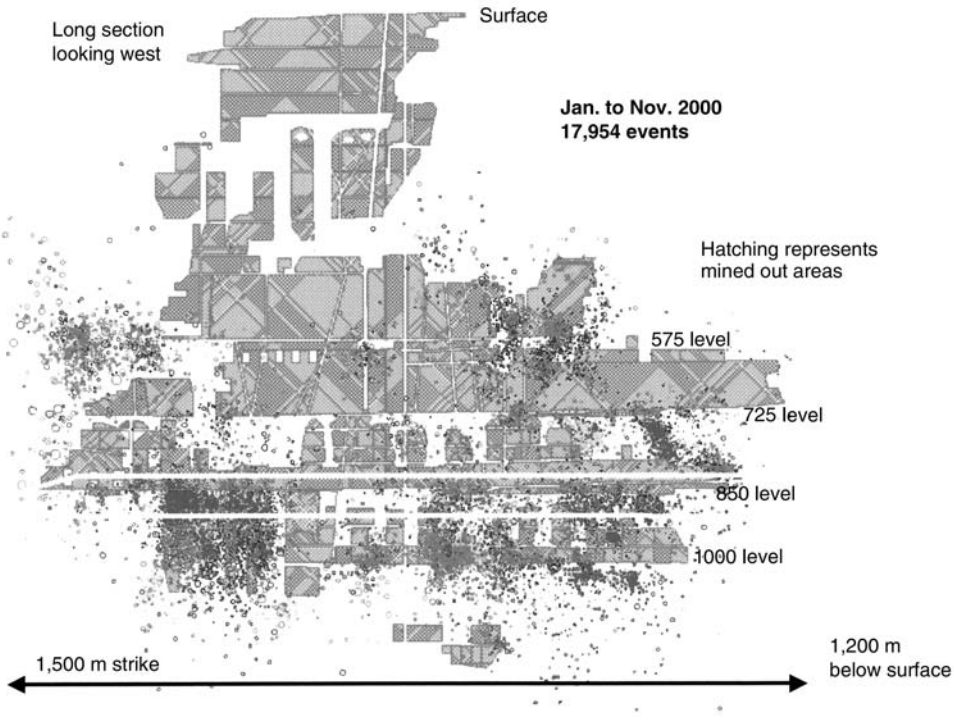


FIGURE 2.1 Brunswick Mine long section

switched mining methods a number of times in order to maintain both safety and production levels. The most recent mining change has been from primary-secondary to pyramidal pillarless mining, as well as two-dimensional pillarless mining fronts in single-lense zones. The mining front is pushed upwards prior to going out sideways to limit back spans to a single mining block. Each mined block is paste-backfilled prior to the mining of an adjacent block, which improves wall and back stability. Stress is transferred away from the stope blocks toward virgin ground, and pillars are not formed.

In addition to a change of mining method, other stress management measures have been implemented. Stress shadowing by mining the hanging wall lense first is carried out where possible, and massive stope destressing has been carried out where applicable. In addition, the ground support in burst-prone areas has been changed from conventional combinations of weldmesh, split sets, cable bolts, rebars, and shotcrete to shotcrete followed by #6 gage chain link mesh, #0 gage heavy mesh straps, and a 1 m $\times$ 1 m pattern of modified cone bolts. This ground support has been found to contain the effects of nearby bursting much better than the conventional ground support previously installed.

Monitoring has also played a critical role in improving ground control and in dealing with rockbursts. The original 32-channel seismic monitoring system has been upgraded to a 96-channel, state-of-the-art, real-time quantitative seismic system. Mining decisions based on microseismic behavior of the wall rocks to mining have become common practice at Brunswick. Real-time nonseismic monitoring, that is, ground movement

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

monitoring and multipoint extensometers, are automatically read through the seismic communication system. A ground control on-call system provides monitoring coverage 24 hours per day, 365 days per year. This is coupled with a systematic preshift analysis of all seismic and nonseismic data recorded over the previous 12 hours. In addition, the miners are instructed to leave any working place if they detect local audible spalling/fracturing or "ground working" and to report these occurrences to ground control from a protected location.

REFERENCES

Anrieux, P., and B. Simser. 2000. Ground stability-based mine design guidelines at Brunswick Mine. In *SME Underground Mining Methods Handbook*. 3rd ed. Edited by W. Hustrulid. Littleton, CO: Society for Mining, Metallurgy, and Exploration, Inc.

Hudyma, M. 1995. Seismicity at Brunswick Mine. *Tenth Ground Control Colloque of the Quebec Mining Association*, Val d'Or, Quebec, Canada.

Simser, B. 2000. Numerical modelling and seismic analysis of events leading up to a violent wall burst. *ISS International Seminar on Modelling with Data*, Stellenbosch, Republic of South Africa.

Simser, B., and P. Andrieux. 2000. Open stope mining strategies at Brunswick Mine. *MassMin 2000*, Brisbane, Australia.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 3

Lake Shore Mine Kirkland Lake, Ontario

SUMMARY

All gold mines in the Kirkland Lake District experienced rockbursts to a certain extent. However, by far the most rockburst-prone mine was Lake Shore, not only in Kirkland Lake but in all of Canada. From 1931 until the mine closed in 1965, 1,319 rockbursts were reported. Some of these bursts were extremely large, with 26 of them greater than 3.0 M_n, including 7 greater than 4.0 M_n. The largest rockburst recorded in Canada, 4.4 M_n, occurred at Lake Shore in September 1939.

Damage from the large rockbursts was also extreme, with levels being completely blocked off over vertical distances up to 500 m. Lake Shore also had 21 rockburst fatalities, which still represents 35% of all rockburst fatalities in Ontario mines.

All types of rockbursts were experienced: strain bursts in shafts, development openings, and stope faces; pillar bursts in sill, remnant, and shaft pillars; and fault-slip bursts on the prominent faults, which included most of the large events greater than 3.5 M_n.

Starting in the mid-1930s Lake Shore expended considerable effort to reduce the rockburst hazard. Central blasting between shifts was initiated. Mining methods were changed from shrinkage to overhand cut-and-fill, then to square set in a longwall rill configuration. Intersecting veins were sequenced to be mined separately. All subsequent shafts were located in the footwall, with no shaft pillars. Concrete was poured for the first and last lift, above and below a level, to limit rockburst damage; steel rings with lagging were used to support access openings in shaft pillars, and rockburst-prone drifts were supported with steel cables with lagging behind them, in a primitive lacing fashion.

GEOLOGY

At Kirkland Lake, narrow gold-bearing quartz veins are associated with a fault system designated as the Main Break, or Kirkland Lake Fault. This fault extends across the whole mining camp in an east-west direction and dips steeply to the south. The Lake Shore Mine is located near the center of the mining camp and has two principal veins, north and south. On the surface the two veins are parallel and are about 120 m apart, extending across the whole property for a strike length of 850 m. On the western side of the mine, the two veins merge at a depth of about 1,200 m.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

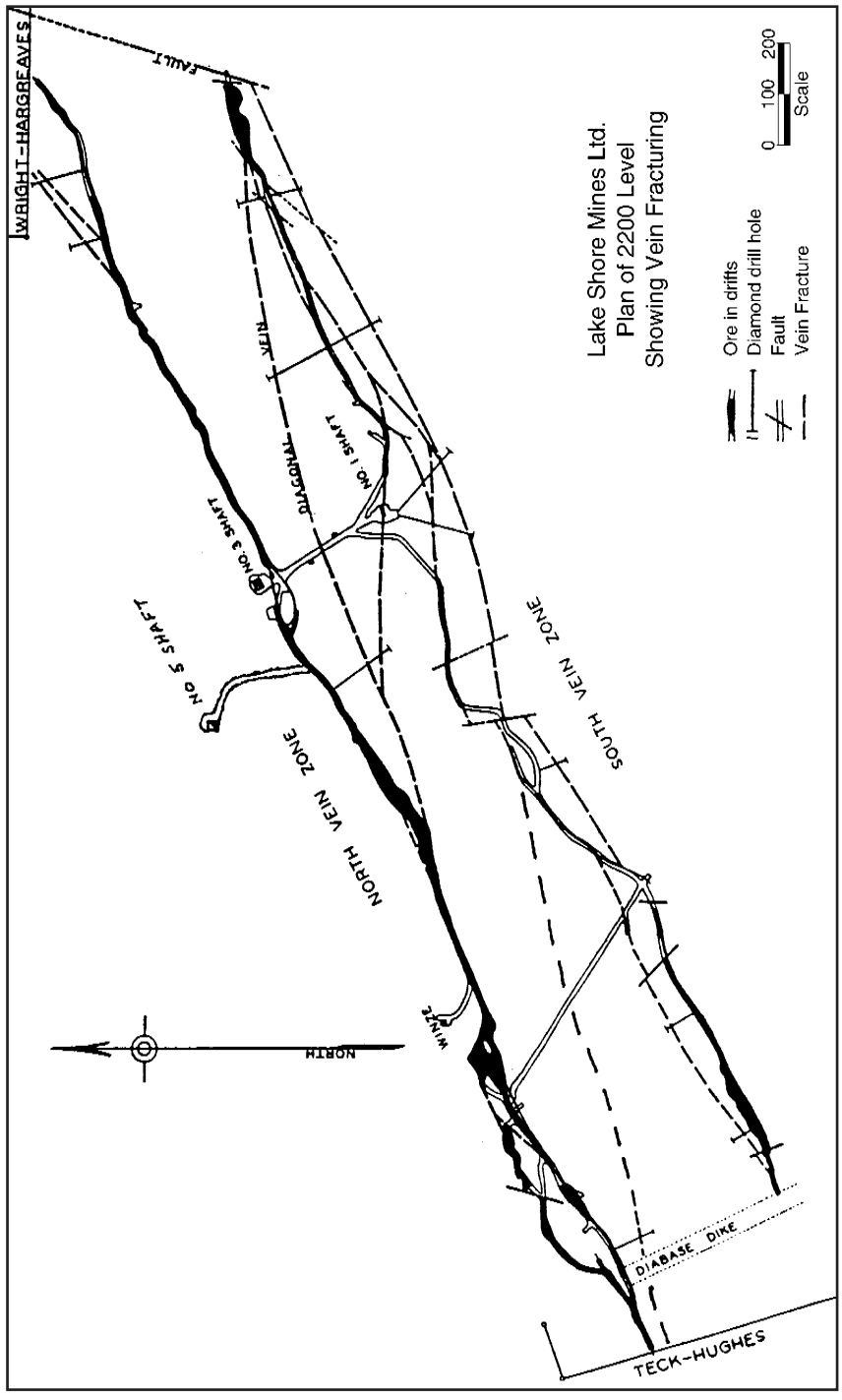


FIGURE 3.1 Plan of the 2200 level, Lake Shore Mine

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

Figure 3.1 is a plan of the 2200 level showing the two ore zones and major structure. The north vein is continuous and, in places, reaches a width of 20 m, although an average of 5 m is more general. The south vein is less continuous and averages 4 m wide. At the western side of the mine, the north vein is the Main Break. Just west of the shafts it deviates along the diagonal vein, which joins up with the south vein at the eastern end of the mine. The south vein then becomes the Main Break to the east. A number of diagonal structures join the north and south veins, some of which are mineralized. There is one prominent transverse fault at the eastern end of the mine, designated the Lake Shore Fault. This fault is almost vertical and has laterally displaced the veins 120 m to the north. From the shafts eastward all the veins are in a porphyry boss, whereas to the west a mixture of porphyry and basic syenite are the host rocks.

Both the north and south veins bottom out near the 6000 level. Ore reappears on the north vein at the 7575 level and was mined down to the 8075 level (2,460 m depth). Rockbursts in these deeper levels, both at Lake Shore and Wright-Hargreaves, are covered in a separate case history.

MINING METHODS

Lake Shore was serviced by three surface shafts and two internal shafts. No. 1 Shaft was sunk in stages to a depth of 1,350 m, in close proximity to the south vein, and it also intersects the north vein at the 3700 level. No. 3 Shaft was located in the immediate hanging wall of the north vein, which it intersects on the 800 level. Below the 1000 level a 45 m wide shaft pillar was left opposite the shaft, which was increased to 90 m wide below the 2000 level, then 120 m wide below the 3075 level. No. 3 Shaft first experienced ground movement problems between the 200 and 600 levels, then severe rockburst problems at greater depths, and had to be abandoned. It was replaced by the No. 5 Shaft, which was sunk entirely in the footwall of the north vein down to the 4000 level. Internal shafts No. 4 and No. 6 were also sunk in the footwall, the former down to the 8075 level, and the latter to the 6075 level. Down to the 2200 level spacing of levels was 60 m, which was reduced to 38 m at greater depths.

Production began at Lake Shore in 1918, and in common with other mines at Kirkland Lake, shrinkage methods were employed. No real problems were encountered with this method down to the 1600 level. In 1929 mill capacity was doubled, which resulted in a rapid pull-down of the broken ore in the shrinkage stopes. This caused massive dilution from the stope walls and ground movements around the shafts. Production from shrinkage stopes was suspended, and years later, this broken shrinkage ore was recovered using square-set methods from the top down.

Below the 1600 level, mining was converted to horizontal cut-and-fill, using waste development rock and alluvial sand as the backfill. No problems were encountered with this method for stopes above the 1800 and 2000 levels. However, the next three lower levels experienced rockbursts when the sill pillars had been reduced to 12 m.

Below the 2700 level, mining was converted to a steep square-set rill system of stoping, using close backfilling. Initially, central raises were driven interconnecting seven levels, generally spaced 120 m or 240 m apart. Rills on the level above were completed before those on the level below could be started. This resulted in a V-shaped longwall configuration. This method with minor modifications was used to mine the remaining ore reserves.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

ROCKBURST INCIDENTS

Starting in the early 1930s, Lake Shore kept meticulous records on all rockburst incidents, which included the date and time of the event. Rockbursts were classified as either *strain*, *light*, *medium*, or *heavy*, depending on the amount and extent of damage and the intensity of the vibrations. Strain and light bursts were localized, medium usually affected more than one level, and heavy affected multiple levels. A written description of the location and amount of damage was accompanied by relevant plans and sections. Any fatalities or injuries were recorded, and sometimes the amount of time to clean up and repair the damage was included. Unlocated bursts were also reported and were invariably classified as heavy, having been felt on surface.

Strain Bursts

Strain bursts were experienced in shafts, development drifts, and stope faces, with damage typically being less than 20 t. No bursts were recorded during sinking of the No. 1, No. 3, and No. 5 Shafts from surface. Both of the internal shafts experienced small rockbursts during sinking. There were 60 recorded bursts in the No. 4 Shaft between depths of 1,340 and 1,570 m, and 30 in the No. 6 Shaft at depths of 1,380 m to 1,540 m. It is interesting to note that there were no further bursts during sinking, especially in the No. 4 Shaft which went down to 2,520 m. Both shafts were sunk entirely within porphyry, and there is no obvious structure that intersects the shafts at the rockburst depths. It is possible that below a depth of about 1,550 m the stress concentrations are high enough to cause the shaft walls to fracture during blasting operations.

Development drifts most affected by bursting were the main crosscut connections to the shafts on each level. Driving these crosscuts was not a problem, but later, when they were subject to stress concentrations due to mining, bursting occurred. Occasionally, isolated drifts would be subjected to bursting, over relatively short distances. In one particular case, a bypass drift on the 4075 level was being driven about 60 m into the footwall of the north vein. About halfway toward the boundary, the drift was turned to eventually intersect the north vein. In this area, over a distance of 40 m, 11 consecutive rockbursts occurred with total damage of more than 400 t. Displaced rock came from the walls and the back, and there was also considerable floor heave. All bursts occurred with or shortly after blasting. This bypass drift had to be lined with concrete over the affected section. These bursts could have been caused by either the change in drift direction relative to the principal stress, a change in rock types, or a combination of both.

In the stopes, strain bursts invariably occurred with blasting and usually only resulted in minimal inconvenience, unless the timber was extensively damaged.

Pillar Bursts

The first pillar bursts were in the sill pillars of the cut-and-fill stopes. These pillars were about 12 m on dip and extended for long distances along strike. The first attempt to deal with this bursting problem was to drive additional raises at 15 m centers, creating a large number of small isolated pillars. This compounded the problem and resulted in multiple pillar failures, sometimes extending over 80 m. Eventually the sill pillars of both shrinkage and cut-and-fill stopes were recovered using square-set methods with backfill. Below the 2700 level no sill pillars were left.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

With the adoption of a square-set rill system of mining, longwalls were established over vertical distances of about 200 m. These longwalls started out at central raises, and then advanced toward each other. This eventually created a triangular shaped regional pillar, with its base on the lowest level. As mining progressed further, the regional pillars became smaller and eventually could be classified as remnant pillars. A number of heavy bursts occurred in these highly stressed remnant pillars. Damage was typically over three levels, with the drifts and rill faces being either completely blocked off or severely damaged. After rehabilitation, mining resumed with no particular problems through the fractured zone, but when solid rock was encountered again, bursting reoccurred in the same pillars. In some cases the rockburst problem became so severe that the remnant pillar was abandoned. In the deeper levels the spacing of the central raises was increased, resulting in the formation of fewer remnant pillars. Below a depth of 1,500 m, ore on the western side became discontinuous, and on the eastern side, longwalls were first established opposite the shafts, then advanced to the east.

The No. 3 Shaft pillar was probably the most common source of rockbursts at Lake Shore Mine. At least 11 major bursts occurred within this pillar, and often they were multiple bursts a few minutes apart. Damage was extensive and often extended hundreds of meters vertically. Undoubtedly, these bursts were caused by slippage along geological structures, as discussed later. However, the pillar was highly stressed and subject to pillar-type bursts. By 1938, No. 3 Shaft had been abandoned and the shaft pillar was no longer required. Mining of this shaft pillar commenced and was still taking place when the mine closed in 1965, with the section between 1400 and 2200 levels being completely recovered. The bursting problem was the same as with the remnant pillars. The edge of the pillar and the levels going though the pillar became highly stressed and burst. After mining the fractured zone, bursting reoccurred. Starting in the 1950s, destress blasting techniques were used for mining of the shaft pillar with mixed results.

Fault-Slip Bursts

Figure 3.2 is a longitudinal section of the north vein. The date and magnitude of the major rockbursts in the shaft pillar are listed. Arrows indicate the approximate center of damage for each rockburst, except for the events in September 1939 and December 1951, where the aerial extent of damage is delineated.

On September 19, 1939, a burst of 4.4 M_n occurred at 10:54 p.m., followed 9 minutes later by a second burst of 3.7 M_n. Over the next 150 minutes there were a further 15 smaller bursts. Damage extended from the 1000 to 3075 levels, over a vertical distance of 630 m and was confined to the shaft pillar. The first burst also caused damage on surface, in the form of broken windows and china, and cracked basement walls. Underground damage was greatest in the north vein drifts, which were blocked off on many levels, and floor heave up to 60 cm was common. In particular, the No. 3 Shaft stations were blocked off on every level between 1400 and 2575. The main crosscuts were damaged from the 1400 to 2825 levels, especially where the diagonal fault intersected those drifts. The south vein drifts were affected between the 2450 and 2950 levels. This burst sequence on September 19 had been preceded by a set of large rockbursts of 3.5 to 4.3 M_n on August 31 and September 2. These earlier bursts had occurred east of the shaft pillar between the 3075 and 3950 levels. The south vein drifts were most affected, with lesser damage on the north and diagonal vein drifts.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

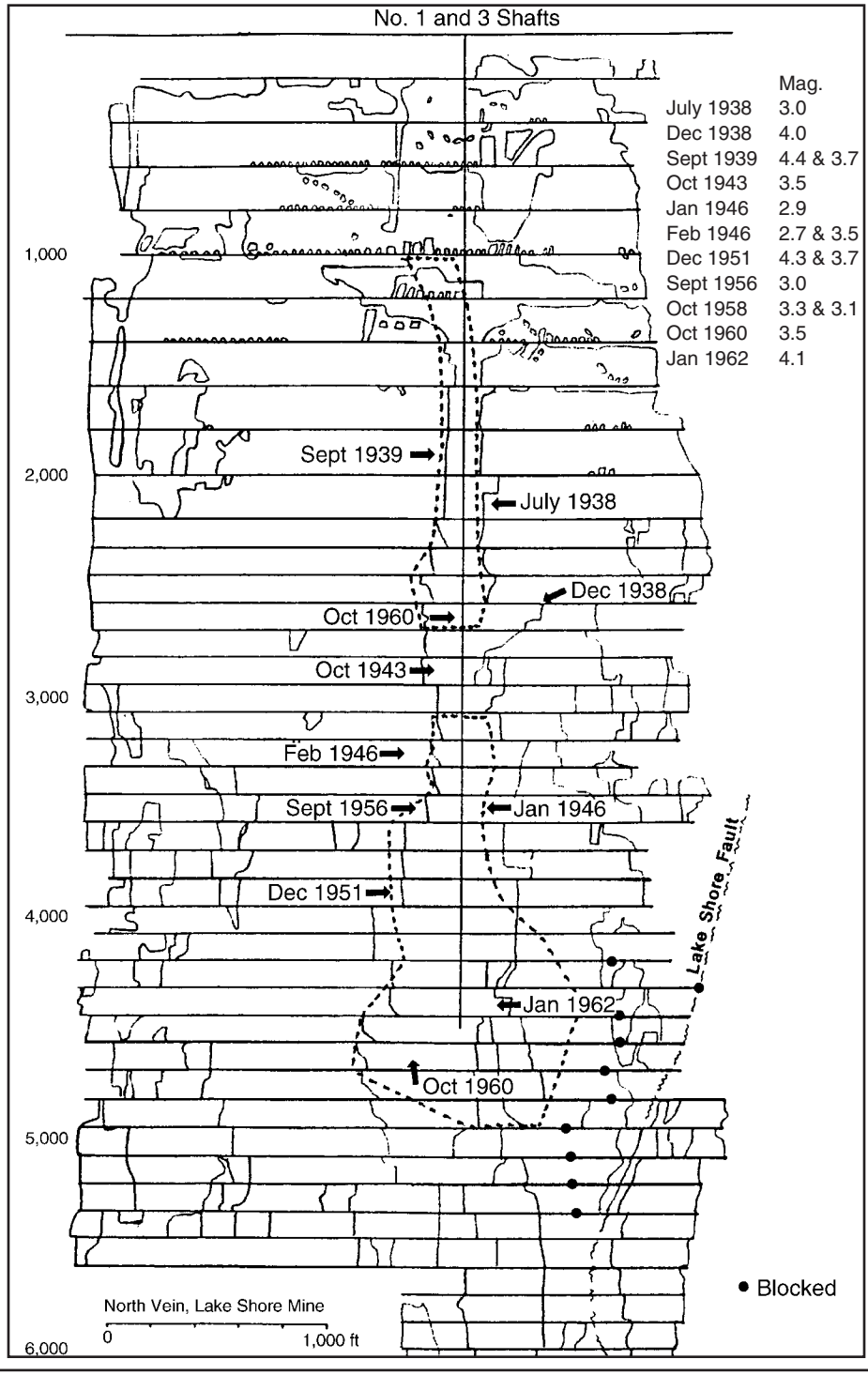


FIGURE 3.2 Longitudinal section showing location of major rockbursts in shaft pillar

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

The second major rockburst sequence in the shaft pillar on December 21, 1951, involved a 4.3 M_n burst at 3:05 p.m., followed by a 3.7 M_n burst. Strong disturbances continued for about 30 minutes, followed by 4 days of seismic activity. Damage extended from the 2950 to 5450 levels, over a vertical distance of 760 m. There were three distinct phases of the damage. From the 2950 to 3200 levels, damage was confined to drifts on the south vein and main crosscuts. From the 3450 to 3950 levels, drifts on the north vein were affected as well as the main crosscuts, which were blocked on the 3450 and 3575 levels. Farther below, in the shaft pillar down to the 4700 level, damage was mainly shake-down of loose rock. From the 4075 level down to the 5325 level, the main damage moved away from the shaft pillar to the east, with drifts on both the north and south veins being blocked near the Lake Shore Fault. It appears that one of the major bursts occurred on the Lake Shore Fault.

Below the 4825 level the north and south veins merge together just west of the No. 4 Shaft and diverge to the east. A vertical longwall, between the 4825 and 6075 levels, was started where the veins merged and advanced eastward toward the Lake Shore Fault. The north vein was mined first, then stopped 20–75 m from the fault. At the same time the extension of both the north and south veins on the eastern side of the Lake Shore Fault were being mined. Mining then commenced on the south vein and had advanced to within 30 to 60 m of the Lake Shore Fault, when a burst of 4.1 M_n occurred in January 1957. The longitudinal section in Figure 3.3 shows the layout of the vein and the locations of damage, which extended from the 5200 to 6075 levels. The stope faces were completely blocked between the 5200 to 5825 levels, and the levels were generally blocked directly below the stope face. Damage extended back about 60 m into the mined-out area on each level. In the north vein, damage was more widespread, extending from the 4950 to 6325 levels. The levels were blocked between the 5450 to 6200 levels, sometimes at the Lake Shore Fault. Damage also extended back into the mined-out areas, as far as the main crosscuts. Due to the blocked off levels, workings to the east of the fault were inaccessible and were abandoned.

The south vein stopes and the levels were rehabilitated and mining resumed 18 months later. These stopes had advanced to within 15 m of the Lake Shore Fault when a second large rockburst of 4.3 M_n occurred in March 1961. The extent and amount of damage was similar to the January 1957 event.

A back analysis has been done, using numerical models, on the September 1939 rockbursts in the shaft pillar and the January 1957 and March 1961 rockbursts on the Lake Shore Fault. With regard to the shaft pillar bursts, the models indicated that the clamping stresses across the pillar would prevent any lateral slippage along the main structures. However, slippage could occur on either side of the shaft pillar on the diagonal fault. This could explain the rockbursts on August 31 and September 2, which did occur in the mined-out workings to the east of the shaft pillar, but not the September 19 events which were solely confined to the shaft pillar. An alternate is that slippage was mainly vertical and occurred on the north vein and diagonal fault. The north and south vein merge at depth forming a wedge of rock. With extensive mining on both sides of the shaft pillar, the weight of this wedge of rock was only being supported by the shaft pillar. The footwall of the north vein was a clay seam. Vertical slippage on this contact then allowed other structures in the hanging wall to slip, including the diagonal fault and south vein.

The back analysis of rockbursts on the Lake Shore Fault is more clear cut. The orientation of this fault is particularly unfavorable to the principal stress direction and large

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

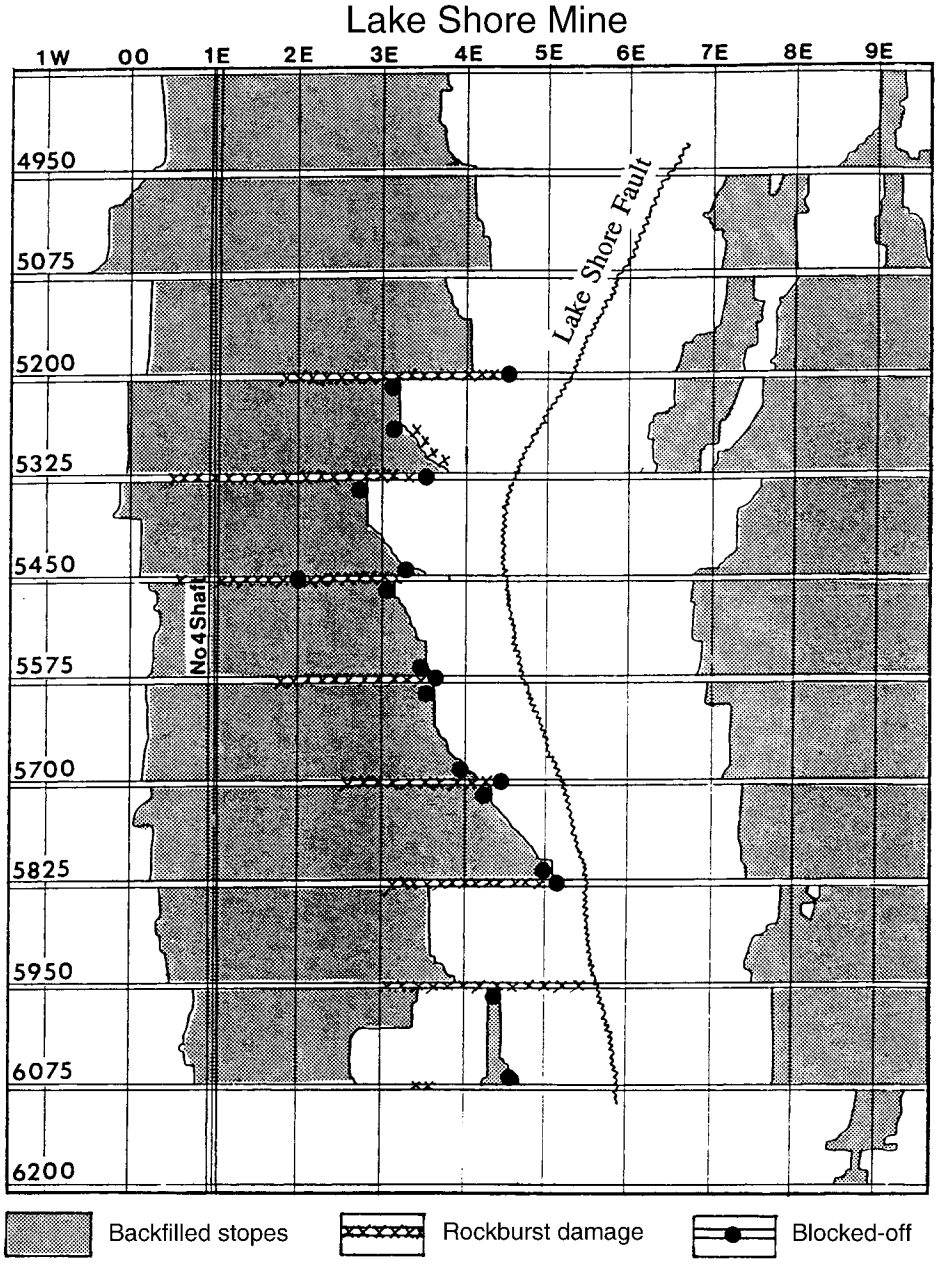


FIGURE 3.3 Longitudinal section of the south vein showing extent of damage from the January 1957, 4.1 M_n rockburst

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

shear stresses can be generated. The models indicate that the January 1957 rockburst most likely occurred on the eastern side of the fault, where extensions of the north and south vein had been mined directly against the fault. Slippage of about 2.2 cm was estimated, with the hanging wall and footwall moving equally toward the stopes. For the March 1961 rockburst, slippage occurred on the western side of the fault, where the south vein intersects the fault. In this case there is greater disparity in the relative movements, with 20 cm displacement of the hanging wall and 10 cm of the footwall, toward the stopes.

DISCUSSION

Of all the mines in Canada, Lake Shore spent the most effort in trying to alleviate their rockburst problems. Tactically, they introduced central blasting between shifts, since most of the strain and pillar bursts occurred with or shortly after blasts. After the rockbursts in the No. 3 Shaft, all subsequent shafts were located well into the footwall, with no shaft pillars. The first lift above a level and the last lift below it were filled with concrete to limit the amount of rockburst damage and clean-up time. They experimented with adding asphalt to the sand fill, to provide some cohesion and yielding characteristics, but this practice was discontinued. In the No. 3 Shaft pillar, drifts in the north vein were supported with circular steel sets with lagging and backfilling. Although these steel sets were considerably distorted by rockbursts in the pillar, some access could usually be maintained.

Strategically, they converted from shrinkage to horizontal cut-and-fill to square-set rill system with backfill. Stope sequence was rigorously controlled to maintain a longwall configuration. This method eliminated the rockburst problem in thin sill pillars. All the old shrinkage stopes were backfilled to improve regional stability. Certain general rules were formulated:

- Mining remnants from two sides is not a good practice.
- Branching vein systems should not be mined concurrently.

Despite all this effort, Lake Shore had the worst rockburst record within Canada and the most rockburst fatalities and injuries within Ontario. This statistic raises the question of what would be done today in mining the Lake Shore orebodies. Certainly cemented paste fill, yielding supports with mesh, and shotcrete would provide better support in the stopes and development drifts. This strategy should alleviate many of the strain bursts and reduce the damage from pillar bursts, but it would have little or no effect on the major fault-slip bursts. Usually, to control the latter, vertical stabilizing pillars are left at set intervals to limit the span of the mine workings and provide more clamping stress on the fault structures. However, Lake Shore did leave a massive vertical pillar around the No. 3 Shaft. Not only did this pillar not act as a stabilizing pillar, it was the source of many of the major rockbursts. On the other hand, the narrow vertical boundary pillars with the Teck-Hughes and Wright-Hargreaves mines were relatively rockburst free. This implies that there was something special about the central area of the Lake Shore Mine.

Only at Lake Shore did the Main Break divert from the north vein at the western side of the mine and cross over to the south vein at the eastern side of the mine. The geologists are of two schools of thought. One group thinks that the cross-over occurred on a

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

single structure, namely the diagonal vein or fault, identified in Figure 3.1. The other group thinks that there are numerous diagonal structures connecting the north and south veins, and that movement occurred on all of them. The latter view is probably more accurate. Under these circumstances, it is virtually impossible to devise a mining method and sequence which would limit or control slippage along these structures.

REFERENCES

Robson, W.T. 1936. Lake Shore geology. *Trans. CIM*. March: 99–141.

Robson, W.T. 1946. Rockburst incidence, research and control measures at Lake Shore Mines. Ltd. *Trans. CIM*. 49:347–376.

Robson, W.T., J.C. Adamson, and W.E. Selnes. 1940. Rockbursts at Lake Shore Mines. *Trans. CIM*. Jan: 7–30.

Robson, W.T., W.E. Selnes, and M.E. Seymore. 1957. Rockburst control measures at Lake Shore Mines, Ltd. *Sixth Commonwealth Mineral and Metallurgical Congress Canada.* 376–384.

Thomson, J.E., et al. 1948. *Geology of the Main Ore Zone of Kirkland Lake.* Vol. LVII. Canada: Ontario Department of Mines. 54–188.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 4

Teck-Hughes and Wright-Hargreaves Mines Kirkland Lake, Ontario

SUMMARY

Teck-Hughes lies directly to the west and Wright-Hargreaves to the east of the Lake Shore Mine. Both mines had a history of rockbursts, dating back from the early 1930s until they closed in the mid-1960s, but to a lesser extent than Lake Shore. Both Teck-Hughes and Wright-Hargreaves placed their main production shafts in the immediate hanging wall of the north vein and left pillars where they intersected or were in close proximity to the vein. All these shaft pillars experienced significant rockburst activity, which damaged the shafts, and in some cases forced them to be abandoned.

GEOLOGY

At Teck-Hughes the north vein, which is also the Main Break, extends across the whole property for a distance of about 450 m. The vein dips 76° to the south and has an average thickness of 2 m but widens out in places to 10 m. Subsidiary veins break away from the north vein into the hanging wall, dipping 40° to 60° to the south. The south vein was not present at Teck-Hughes. No major geological structures intersect the ore zone. Host rock is mainly syenite porphyry with some tuff inclusions.

At Wright-Hargreaves there were 10 vein structures, the main ones being the north and south veins, the latter being the Main Break. The two veins are roughly parallel to each other, some 120 m to 150 m apart, and dip steeply to the south. The subsidiary veins are either subparallel to the main veins or inclined between the north and south veins. East of the Lake Shore Fault, the ore zone is intersected by a number of subhorizontal strike faults and steeply dipping cross faults that have displaced the veins by up to 200 m. The faults separated the mine into a series of semi-isolated blocks. Down to a depth of 2,000 m the host rock was syenite porphyry, but below this level there were inclusions of conglomerate, tuff, and granite porphyry.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

MINING METHODS

Teck-Hughes was serviced by two main shafts from surface, both being sunk in the hanging wall of the north vein. The central shaft extends to a depth of 910 m and intersects the north vein on 16 level, at a depth of 560 m. An irregular shaft pillar, 30 m to 75 m wide by 105 m long, was left between 16 and 19 levels. The south shaft and its extension were sunk to a depth of 1,660 m. The extension shaft intersects the north vein on 34 level, at a depth of 1,240 m. Again, an irregular shaft pillar about 75 m wide by 110 m long was left between 33 and 36 levels. The lowest levels, down to a depth of 1,860 m, were serviced by internal inclined winzes. Levels were established at 38 m intervals.

Shrinkage stoping was used almost exclusively at Teck-Hughes. Regular backfilling of stopes was not done, except for some stopes near the shafts, which were filled with waste rock. The walls of most shrinkage stopes were allowed to slough, until they were completely filled. This increased the original stope size by a factor of about eight. In subsequent years some of these shrinkage stopes were pulled for a second time, resulting in even larger openings. Sill pillars of the shrinkage stopes were recovered either by vertical slices or in a rill configuration.

All major shafts and infrastructure at Wright-Hargreaves were located in and between the north and south veins. No. 3 and No. 4 Shafts extended down to a depth of 1,230 m, the former in close proximity to the south vein, and the latter in the hanging wall of the north vein. No systematic shaft pillars were left to protect these shafts. Rather, the policy was to leave pillars where the veins were within 20 m of the shafts, otherwise they were mined out. One particular 50-m-wide pillar was left in the north vein between the 3900 and 4050 levels, directly opposite and within 3 m of No. 4 Shaft. Deeper levels were serviced by two internal shafts; No. 5 down to a depth of 1,950 m and No. 6 to 2,470 m. Levels were established at 47 m intervals.

Up to 1930 all stoping in the upper levels at Wright-Hargreaves was done by shrinkage techniques. Then, cut-and-fill rill stoping was introduced and remained the main mining method until the mine was closed in 1965. However, some modified shrinkage methods continued to be used in narrow stopes. Timber stulls were installed between hanging wall and footwall to provide extra support. With increasing depth, larger sill pillars, up to 25 m wide, were left above both cut-and-fill and shrinkage stopes. These were recovered using vertical slices.

ROCKBURST INCIDENTS

From the early 1930s to the mid-1960s, Teck-Hughes reported 150 rockbursts, of which 40 were classified as heavy; and Wright-Hargreaves reported 230 bursts, with 23 being classified as heavy. However, neither mine reported small strain bursts if no injury occurred, nor unlocated bursts if there was no damage reported in active mine workings.

Teck-Hughes Mine

Figure 4.1 is a longitudinal section of the Teck-Hughes Mine, showing the mining layout of 1939 and the locations of 26 rockbursts that occurred between 1936 and 1939. There are two areas of burst concentrations—around the west side of the 38 level and the central area of 45 level. Around the 38 level the bursts mainly occurred in the thin sill pillars at the top of the shrinkage stopes. This area was also within a very brittle tuff inclusion. Around the 45 level the mine was experimenting with driving sublevels between the

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

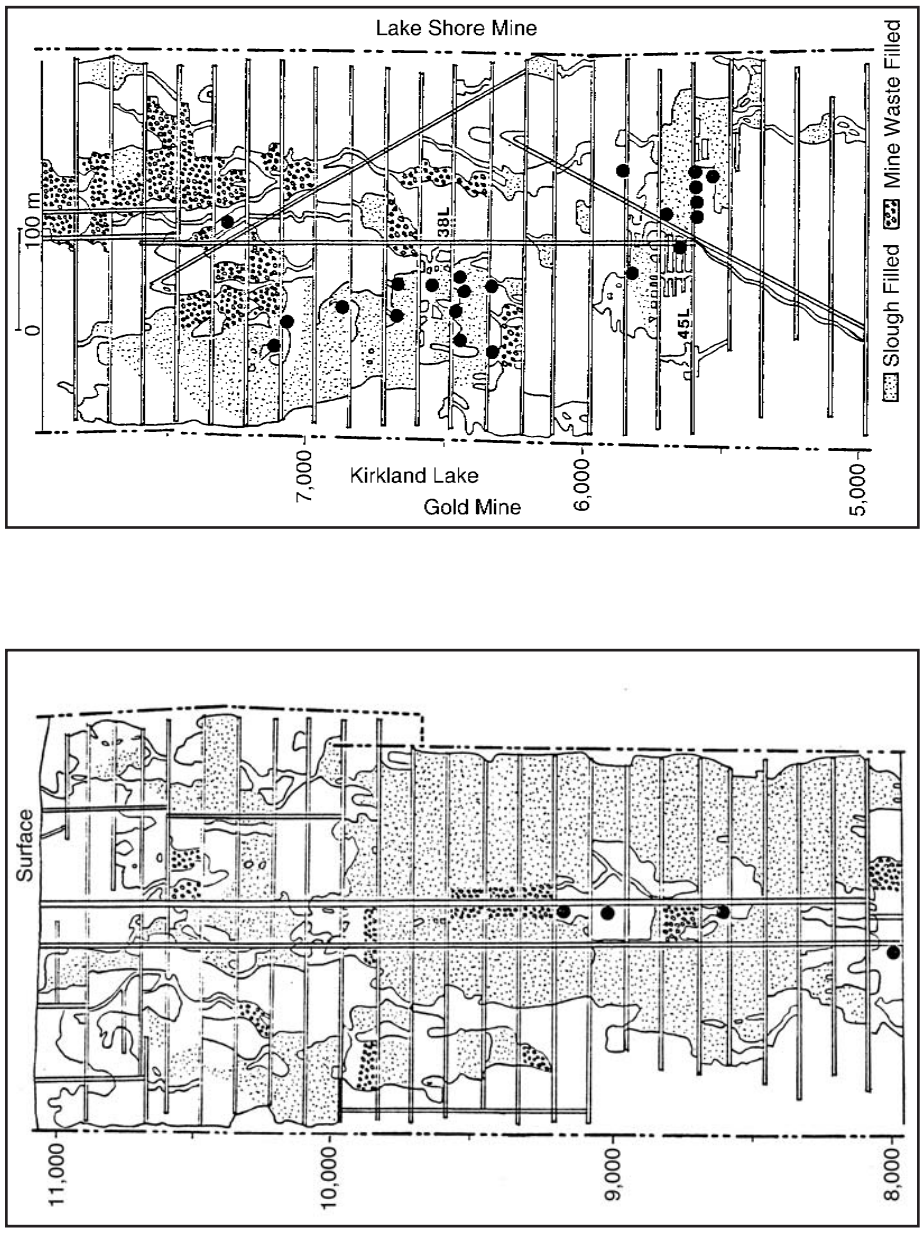


FIGURE 4.1 Mining layout at the Teck-Hughes Mine showing locations of rockbursts (after Christian 1939)

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

main levels, creating a series of thin sill pillars which became overloaded and burst. It was realized that this was not good mining practice.

Of particular interest are the rockbursts that occurred in the shaft pillars. In April 1936 a heavy burst occurred in the central shaft pillar between the 17 and 18 levels. Both footwall and hanging wall sides of the shaft were damaged over a vertical distance of 47 m. In October of the following year, a burst occurred just below the 16 level, which caused 0.6 m of floor heave in the shaft station and damaged the shaft for 12 m below the level. In February 1955 three rockbursts occurred in rapid succession, which caused damage to shaft timbers between the 17 and 19 levels. The damage from all these rockbursts was repaired and the central shaft continued in operation.

The south shaft extension first experienced bursts in June 1957, when the 38 level shaft station was slightly damaged. In August 1958 a larger burst caused damage to the shaft timbers and sets between the 35 and 36 levels. In November of the same year two rockbursts, both of 3.1 M_n, occurred 15 minutes apart. Extensive damage of more than 300 t was done between the 33 and 36 levels, over a vertical distance of 115 m. As a consequence, the south shaft extension was abandoned as a hoisting shaft between the 30 and 36 levels and bulkheaded above the 36 level.

Wright-Hargreaves Mine

Initially the rockburst problem at Wright-Hargreaves was not as severe as at the Lake Shore and Teck-Hughes mines. The strike and cross faults separated the mine into a series of semi-isolated blocks. Between 1932 and 1956 reported rockbursts averaged five per year. However, toward the end of the mine's life, extraction was mainly from pillars and remnants, and regional stability became increasingly dependent upon the stability of the faults. As a result, between 1957 and 1964 reported rockbursts increased to an average of 15 per year.

Almost all of the rockbursts at the Wright-Hargreaves Mine were confined to the north vein. Strain bursts occurred in development openings and stope faces, and pillar bursts in the thin sill pillars. In the latter case the size of the pillar was increased and recovered using vertical or rill slices. Also, all the old shrinkage stopes above the 3900 level were sand filled. Figure 4.2 is a longitudinal and cross section through the central part of the mine. In 1964 mining was initiated to recover the shaft pillar between the 3900 and 4050 levels. Previously an inclined waste pass had been driven through the central part of this pillar. Mining began at this inclined waste pass using shrinkage. Stope width was 2 m to 3 m and the walls were supported by 300 mm diameter stulls on 1.2 m centers.

On August 14, 1964, the day shift blasted 70 holes in this stope with no accompanying signs of undue pressure. However, at 9:27 p.m. a 3.1 M_n rockburst occurred causing the stope to collapse, resulting in two fatalities. The levels directly above and below the stope also collapsed, and the access drifts to these levels were blocked. In addition, the sump of the No. 4 Shaft was completely filled with broken rock up to the 3900 level, and there was considerable damage to the shaft timbers above this level.

Seismic activity continued and was so severe that it was decided to evacuate the mine. During this evacuation a major rockburst of 4.2 M_n occurred at 11:52 p.m. This burst caused major damage from the 2550 to 3750 levels, as indicated on the sections in Figure 4.2. On each level the main crosscut from No. 4 Shaft was blocked at or near where it intersects the north vein. The lateral extent of damage was never established

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

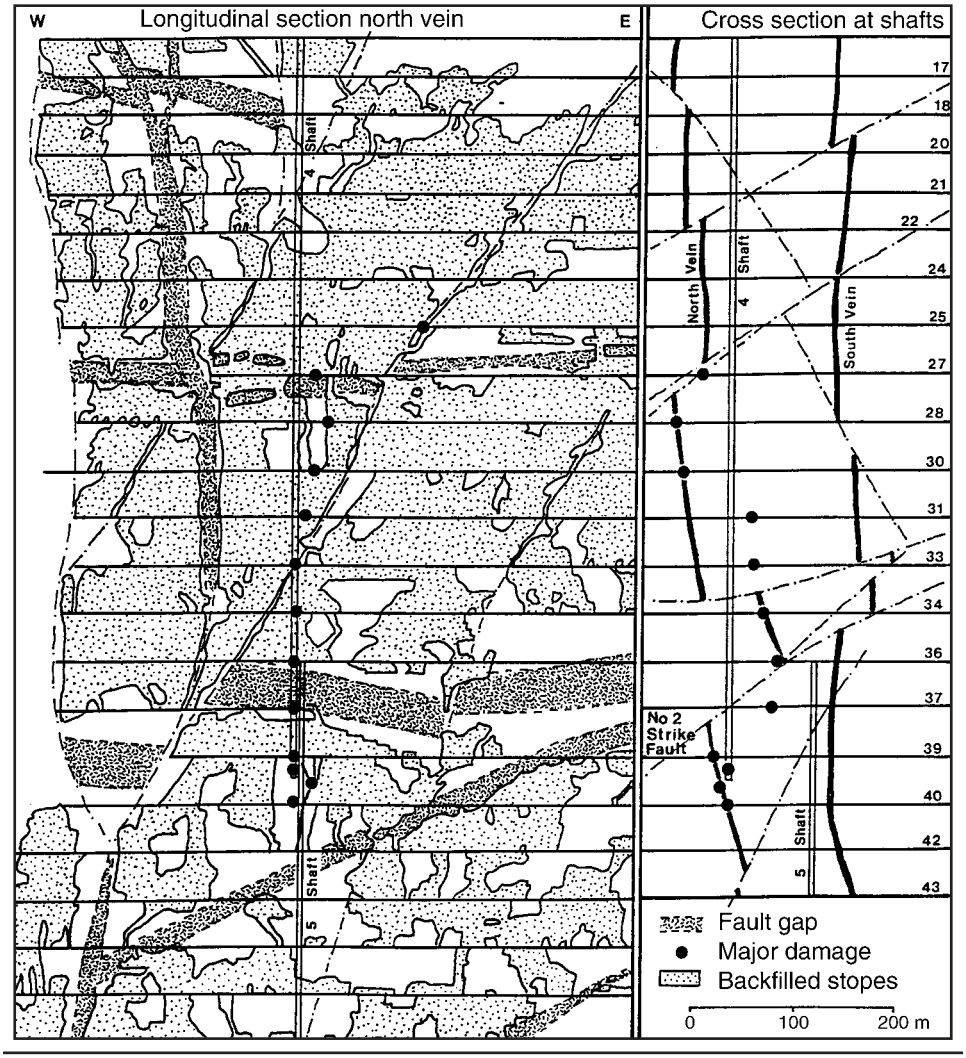


FIGURE 4.2 Longitudinal and cross sections of the Wright-Hargreaves Mine showing extent of damage from August 1964 rockbursts

but must have involved thousands of tonnes. The No. 5 Shaft was also affected and on the 3600 level the drums of the hoist were displaced 150 mm.

The first large burst of 3.1 M_n was a typical pillar burst, as damage was localized and confined to the shaft pillar. The second and larger burst of 4.2 M_n had to be caused by slippage along a geological structure, because damage was extensive over 365 m vertically. From the cross section the most likely structure is the No. 2 strike fault, which dips about 30° to the north. Due to the damage to the No. 4 Shaft and limited ore reserves, the mine was put on a salvage basis and closed in March 1965.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

DISCUSSION

Both Teck-Hughes and Wright-Hargreaves state that by the time rockbursts became a serious problem in the mid-1930s, all their main shafts were already in place. They were invariably in the hanging wall, leaving relatively small pillars where they intersected, or were in close proximity to the veins. Subsequently, all these shaft pillars experienced significant rockburst activity. In hindsight, both mines recognized that it would have been better to either mine these pillars at an early stage and backfill them or to locate the shafts entirely in the footwall. The latter solution was considered too costly. However, as R.G.K. Morrison pointed out in his report on rockbursts in Ontario mines, the extra cost of footwall crosscutting was small when compared to what happens to shafts protected by inadequate pillars.

Both mines had radically different views on the use of backfill to reduce the rockburst hazard. Teck-Hughes thought that backfill had no effect on the incidence of bursts and that close filling was mainly a matter of economics. Whereas Wright-Hargreaves believed that tight filling of all openings, as soon as possible, was required to provide resistance to stope wall closure and to limit the formation of domes. In some respects, both views were valid. The alluvial sand and waste rock fill being used were very compressible and, hence, provided little resistance to movement of the stope walls. However, at depths greater than 2,000 m convergence of the stope walls is significant. This compresses the fill to a point where it transmits load, which otherwise would have to be carried by the pillars.

In most mines, recovery of the initial 70% to 80% of the ore reserves can normally be carried out without undue ground control problems. The remaining 20% to 30%, and especially the last 5%, are always more difficult to mine due to high-stress concentrations. This is compounded when small remnant pillars have been left near the center of the mine workings. In retrospect, it would have been better for Wright-Hargreaves to have abandoned the small ore reserves in the shaft pillar between the 3900 and 4050 levels, especially since the shaft was still being used.

REFERENCES

Buckle, F. 1965. The rockburst hazard in Wright-Hargreaves Mine at Kirkland Lake, Ontario. *Canadian Mining Journal.* 86(9):81–87.

Christian, J.D. 1939. Rockbursts at Teck-Hughes Mine. *Trans. CIM*. 42:550–567.

Hopkins, H. 1940. Faulting at the Wright-Hargreaves Mine with notes on ground movement. *Trans. CIM*. 43:685–707.

Morrison, R.G.K. 1942. Report on the rockburst situation in Ontario Mines. *Trans. CIM*. 43:225–272.

Robertson, A.F. 1939. Rockbursts at Wright-Hargreaves Mine. *Trans. CIM*. 42:583–592.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 5

Lake Shore and Wright-Hargreaves Mines at Depth

SUMMARY

At Wright-Hargreaves and Lake Shore, the lower levels are separated from the upper mine workings by either a fault or waste gap. The lower levels were mined to a depth of 2,470 m, which still represents the deepest mine workings in Canada. Rockbursts occurred in both mines, in their deeper workings, but not to the extent that could be anticipated from the increased depth. Only one heavy burst occurred at Lake Shore and one rockburst of 3.2 M_n at Wright-Hargreaves. Mainly, the bursts were of the strain type, which usually occurred with or shortly after blasting and caused minimal disruption. Of interest is that the remnant pillars, produced by the rill longwalls at Lake Shore, were not rockburst-prone, compared to the severe rockburst problem in remnant pillars in the upper levels.

GEOLOGY

Figure 5.1 shows a composite geological plan of the deepest levels at Lake Shore and Wright-Hargreaves. East of No. 4 Shaft at Lake Shore the north vein is planar, striking N30°E, dipping 80° to the south, and has an average width of 2.3 m. Lake Shore also mined the No. 4 Vein, lying 30 m to 150 m south and in the hanging wall of the north vein. At Wright-Hargreaves the north vein persists, but there are also two subparallel veins which are mineralized intermittently in the hanging wall.

At Lake Shore both veins are totally within syenite porphyry. On crossing over the boundary with Wright-Hargreaves, the footwall of the north vein becomes tuff, then conglomerate. Around No. 6 Winze a granite porphyry dyke intersects the vein structures.

In the upper levels the most prominent transverse fault is the Lake Shore Fault, which has laterally displaced the veins. Between the 6325 and 6575 levels the No. 5 strike fault from Wright-Hargreaves merges with the Lake Shore Fault, and just below this level the Lake Shore Fault merges with the north vein, east of the No. 4 Shaft. Both the vein and the fault occupy the same zone down to the 8075 level and beyond.

West of the No. 4 Shaft the north vein is the Main Break, which crosses over to the south vein opposite the shaft. This structure was barren below the 6075 level.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

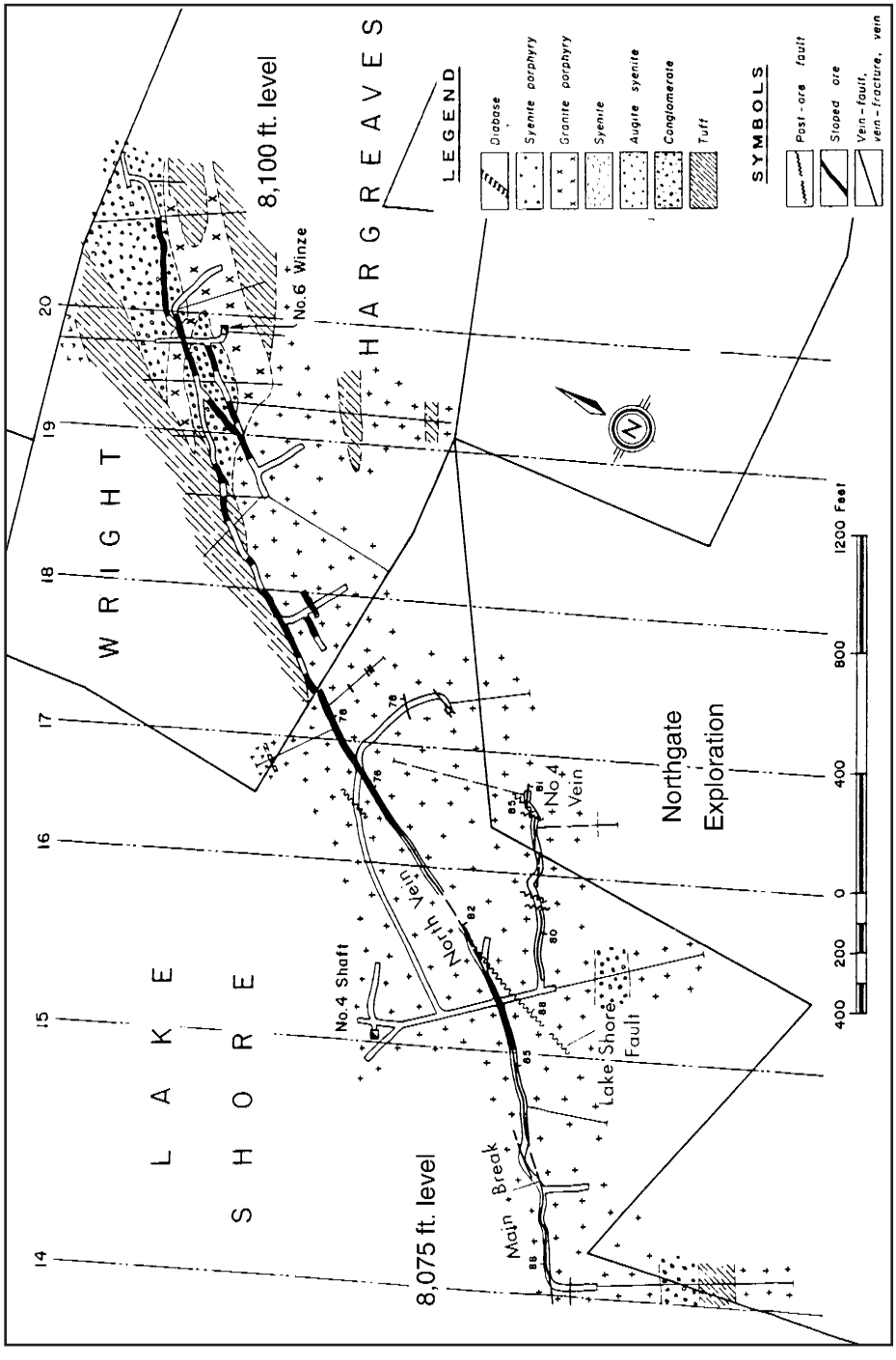


FIGURE 5.1 Composite geological plan of the deepest levels at Lake Shore and Wright-Hargreaves mines

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

MINING METHODS

At Lake Shore, ore on the north vein was continuous down to the 5450 level, then became sporadic and bottomed out on the 6325 level. Ore reappeared at the top of the 7575 level and was mined down to the 8075 level, by extending the No. 4 internal shaft in the footwall. At Wright-Hargreaves, the north vein is cut off by the No. 5 strike fault at the 5200 level. There is a fault gap of about 150 m and ore reappeared on the 5700 level. This ore was mined down to the 8100 level, via the No. 5 and No. 6 Winzes.

Lake Shore used their square-set rill system of stoping with close backfill. In this case an inverted-V longwall configuration was used, with the bottom stopes leading. Initially, central raises were driven between the 8075 and 7575 levels, spaced about 120 m apart. Mining then proceeded from these raises in an east and west direction. This resulted in the formation of wedge-shaped remnant pillars, against the boundary with Wright-Hargreaves and near the center of the mining zone. Destress blasting techniques were used in these remnant pillars. With the longwall method, areas of low grade were mined to maintain the stoping line.

At depth Wright-Hargreaves mainly used overhand cut-and-fill methods, but at least one shrinkage stope was mined from the 7800 level. In both methods the stopes were stopped 25 m below the level and converted to either vertical or rill slices. It appears Wright-Hargreaves used a cut-off grade to determine mineable ore, which resulted in a number of scattered stopes and pillars.

Both mines used alluvial sand and waste development rock for backfill. However, when development work had been completed, probably only alluvial sand was used. Lake Shore continued their practice of pouring concrete for the cuts above and below each level.

ROCKBURST INCIDENTS

Figure 5.2 shows a composite longitudinal section of the mine workings below a depth of 2,000 m. Stopes in the three ore zones at Wright-Hargreaves are superimposed, whereas the workings in the No. 4 Vein at Lake Shore are shown in the insert diagram. The scattered stoping at Wright-Hargreaves and the longwall mining at Lake Shore are self-evident. Over a 10-year period, 34 rockbursts were reported by Lake Shore and 15 by Wright-Hargreaves for these deep workings. The locations of these rockbursts are also plotted in Figure 5.2.

There is a high concentration of rockbursts along the longwall advancing toward the boundary with Wright-Hargreaves and just over the boundary. At first glance, it would appear that these bursts were caused by the high stress associated with an advancing longwall mining geometry. However, other longwalls advancing toward the west and toward each other at the center of the mining zone were not rockburst prone. The differentiating feature of the boundary area is that the hard and brittle tuff forms or is in close proximity to the footwall contact.

A noticeable feature at both mines is that 70% of the rockbursts at Lake Shore and 80% at Wright-Hargreaves occurred with a blast, between shifts, or over the weekend. This is a higher percentage than was reported during mining in the upper levels. Again at both mines there were minor rockburst-related injuries, involving less than a tonne each, but no rockburst fatalities.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

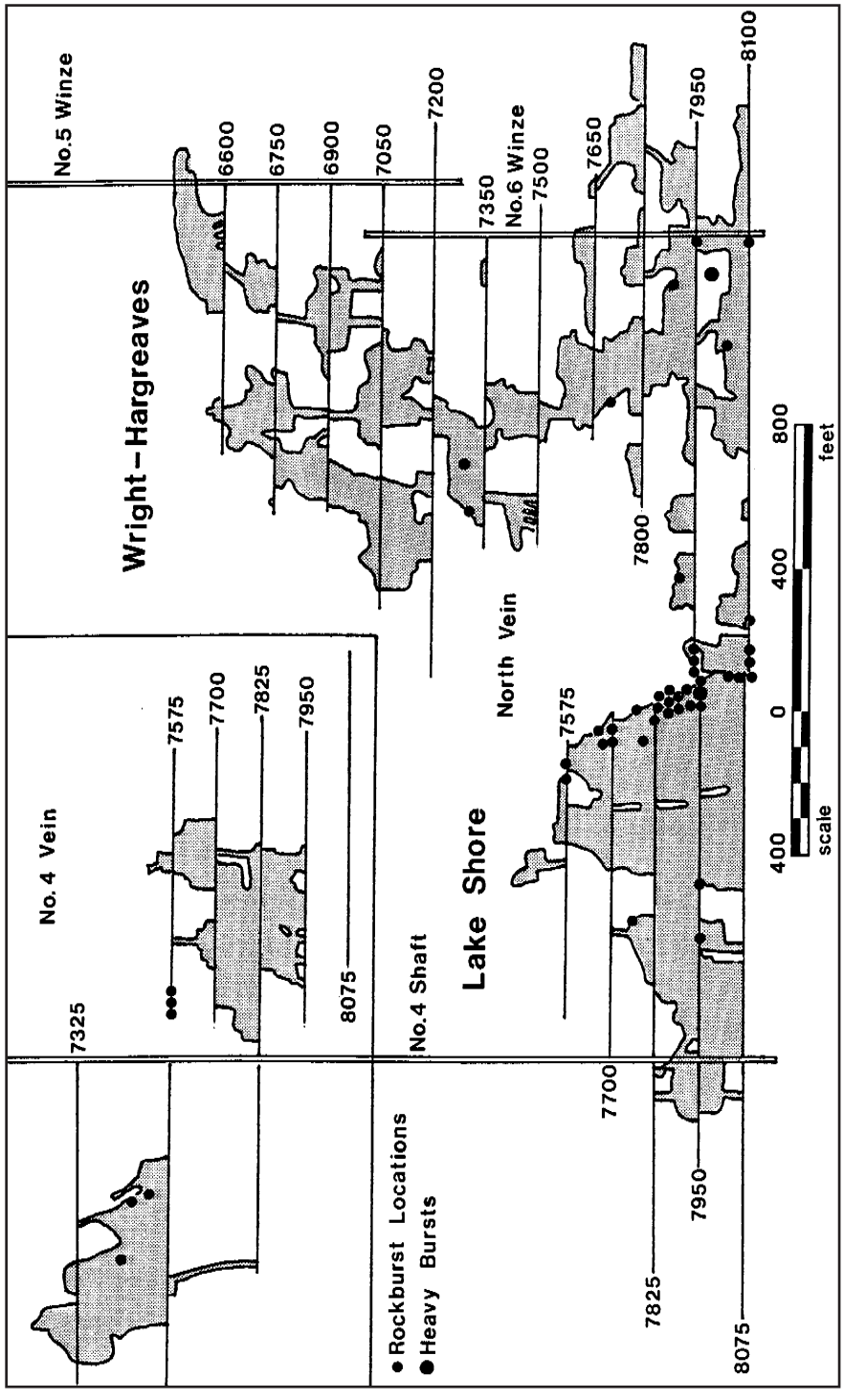


FIGURE 5.2 Composite longitudinal section showing rockburst locations at Lake Shore and Wright-Hargreaves mines

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

Of the 34 rockbursts at Lake Shore, 25 of them were classified as "strain," typically involving about 20 t of damage. Eight bursts were classified as "light" involving 25–100 t of damage. The one heavy burst occurred on February 27, 1963, just after a blast and about 30 m from the boundary. The 7950 level was blocked, as were the stopes directly above and below. Seismograph records in Ottawa were checked for this burst, but no trace could be found. Hence, it was less than 3.0 M_n, the level of detectability for Kirkland Lake at that time.

About half of the rockbursts at Wright-Hargreaves occurred in sill pillars, and in some cases, the damage was also larger, involving more than 150 t. On June 26, 1964, a burst of 3.2 M_n occurred just after a blast. It was centered just west of No. 6 Winze where there are three interconnected veins. In this area the host rock was a relatively weak conglomerate, surrounded by a strong and brittle granite porphyry. An area 60 m long by 30 m wide was blocked off, including the 7950 and 8100 levels. As a result these two lower levels were abandoned.

DISCUSSION

The rockburst activity, especially at Lake Shore, is not commensurate with the increase in depth. Inspection of rockburst, production, and depth of mining records at Lake Shore reveals that rockburst activity did increase with depth down to 1,200 m below surface. At greater depths, down to 2,460 m, rockburst activity was relatively constant and was controlled by the rate of production. Between 1937 and 1965, when mining was concentrated between depths of 1,200 m to 2,460 m, damaging rockbursts averaged 1 per 7,400 t mined.

Rockburst activity at Wright-Hargreaves was relatively low on a yearly basis, except for the last 8 years. During this latter period, mining was concentrated in the deepest levels, as well as recovery of pillars and remnants in the upper levels. Damaging rockbursts averaged 1 per 10,000 t mined for this period, which is close to the rate of damaging bursts at Lake Shore.

An interesting feature at Lake Shore in the deeper levels is the lack of very large magnitude bursts, which would cause damage to multiple levels. In part this can be attributed to the lack of major structures intersecting the ore veins. In the upper levels the transverse Lake Shore Fault was a major hazard where it intersected the vein. Whereas in the deepest levels, the fault became part of the ore zone. Also, the Main Break separates from the north vein to the west of the mine workings and caused no burst problems.

Another interesting feature at Lake Shore was the absence of rockbursts in the remnant pillars near the center of the ore zone. These pillars, as well as the advancing longwall to the east, were destressed using 10-m- to 20-m-long diamond drill holes spaced on 8 m centers. At this spacing, destress blasting was probably ineffective, and it was noted that no bursts were ever triggered by these blasts at Lake Shore. However, it should be pointed out that these types of "springer" destress holes were commonly used in burst-prone mines, and both miners and management swore by them. It was also noted that in the central area containing the remnant pillars, stopes had closed from their original width of 1.5 to 1.8 m to 0.7 m. Even loose alluvial sand would exhibit strain-hardening characteristics with this amount of closure, and would have a significant load bearing

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

capacity. It appears that the remnant pillars eventually crushed and became too difficult to mine, as Figure 5.2 indicates that thin rib-type pillars were left instead.

REFERENCES

Buckle, F. 1965. The rockburst hazard in Wright-Hargreaves Mine at Kirkland Lake, Ontario. *Canadian Mining Journal.* 86(9):81–87.

Charlewood, G.H. 1964. *Geology of Deep Developments on the Main Ore Zone at Kirkland Lake.* Geological Circular No. 11. Canada: Ontario Department of Mines.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 6

Macassa Mine Kirkland Lake, Ontario

SUMMARY

Rockbursting did not become a serious problem at Macassa until almost 50 years of mining had been carried out. However, once the deeper and more extensive ore zones in the western part of the mine were mined, large damaging bursts were induced. Mining methods were changed to deal with this problem, and rock mechanics input into mining was increased, but bursting continued to be a serious problem. The rockburst sequence in April 1997 put the No. 3 Shaft out of service, led to mining being restricted to above the 5300 level, and resulted in a premature closure of Macassa in 1999. It should be noted that Macassa was reopened in late 2002 with new owners.

GEOLOGY

At Kirkland Lake the gold-bearing ore zones are associated with a prominent fault system known as the Main Break. At the No. 3 Shaft area of the Macassa Mine, the main ore-bearing structure is the 04 Break, which is subparallel to the Main Break. The 04 Break is really a parallel series of structures that vary in width from 0.6 m to about 18 m. Above the 5600 level, some 1,700 m in depth, the 04 Break typically is the footwall of the mineralization. Below the 5600 level the 04 Break forms the hanging wall of the mineralization, with a subparallel North Break defining the footwall in localized areas. There are numerous splays and splits off of these major structures, as well as parallel structures in both the hanging and footwall. The 04 Break strikes an average N65°E and dips 70° to 75° to the south. The gold-bearing veins occur in quartz-filled fractures associated with the pre-ore faults and fractures. The overall strike length of the ore zones at Macassa is about 3,660 m. The western plunge of the orebody has resulted in mineralization to depths greater than 2,210 m at the western end of the mine. At Macassa the mineralization is not continuous, as was the case at Lake Shore and Wright-Hargreaves. This has resulted in waste pillars in all of the ore zones.

Besides the pre-ore structures along the vein, there are a number of post-ore structures that cross and offset the mineralization along the 04 Break. The Amikougami Creek Fault causes left lateral movement of the 04 Break and cuts off both the 04 Break and mineralization about 275 m west of the No. 3 Shaft. The Tegren Fault is subparallel to the Amikougami and causes similar left lateral displacement of the 04 Break to the

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

north, separating the west ore block from the east ore block in the No. 3 Shaft area. Both these faults dip steeply to the east. The 04 North Break is a shallow dipping fault that displaces the east ore block to the south.

The three major rock units at Macassa are tuff, basic syenite, and syenite porphyry. The tuff and syenite porphyry are stiffer and more brittle than the basic syenite. The spatial distribution of these rock types is very complex, but in general, the stratigraphic units and the long axis of all the intrusives strike N60° to N80°E and dip steeply to the south. Below the 6600 level in the west ore block the wall rock is primarily basic syenite.

MINING METHODS

Production at Macassa began in 1933 near the eastern limit of the property. Mining was carried out from the No. 1 Shaft, which was sunk to 686 m. To service deeper mining the No. 1 Winze was sunk to 1,448 m. As mining progressed to the west, the No. 2 Shaft was sunk to 1,410 m, followed by the No. 2 Winze to about 2,100 m. The lowest mining level was the 6450. Initial mining was carried out by shrinkage methods. At depth the mine was converted to overhand cut-and-fill with development waste rock used for fill. Mucking in stopes was done with slushers, and production gradually increased to about 450 t per day.

Production was both limited and difficult owing to the large lateral extent of the mine to the west and the three-tiered hoisting system. In addition, ore reserves at the western limit of the property were projected to below 2,100 m, hence, a decision was made to sink the No. 3 Shaft at the west end of the mine from surface down to 2,204 m. This would allow mining down to the 7050 level. The No. 3 Shaft, which intersected the 04 Break at the 5300 level, is timbered with a 12 ft by 28 ft section. It is oriented with its long axis perpendicular to the vein. This shaft was completed in 36 months and was put into service in 1986. Seismic activity first occurred during shaft sinking at a depth of 400 m. Below 1,020 m small rockbursts became a continuous problem and wall destressing had to be carried out. The worst bursting was in the northwest and southeast corners of the shaft in the tuff formations. The largest burst displaced some 35 t of rock and threw the shaft crew as far as 7.5 m across the shaft. A 61-m-wide shaft pillar was left from the 5125 level down to the 5725 level. The shaft was planned to intersect the orebody, because it was felt that too much drifting to reach the ore would result from siting the shaft in the footwall.

After the No. 3 Shaft was put into service, the waste backfill was changed to a cemented rock fill. Instead of leaving sill pillars just below the level, the pillar was placed in the center of a 61 m high stoping block between levels. Hence, a stope was benched down from the level above, using underhand cut-and-fill, and mined up from the level below using overhand cut-and-fill. The rock fill was from a nearby surface quarry. The purpose of placing the pillar in the center of the stope was to limit the damage from pillar bursting, which was a serious problem for Macassa at depth. It was also anticipated that a stiff cemented rock fill would restrict closure and reduce the incidence of bursting. Bursting continued despite the use of cemented rock fill. Figure 6.1 is a long section of the western part of the mine.

Mining on the lower levels at Macassa was stopped in 1997 following a rockburst sequence that put the No. 3 Shaft out of service below the 5700 level. Mining above the 5300 level continued, but the mine was closed in mid-1999 due to falling gold prices.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

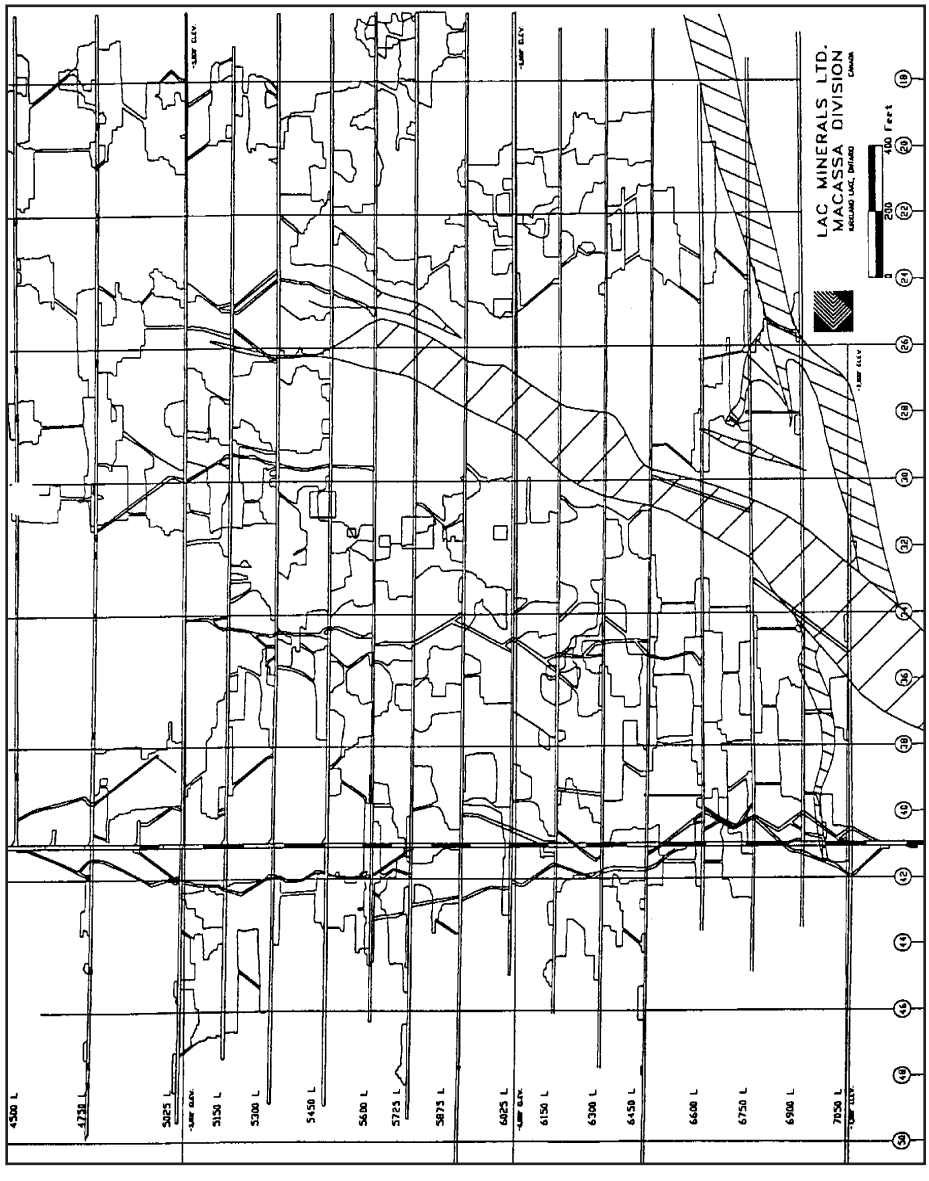


FIGURE 6.1 Longitudinal section—Macassa Mine

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

ROCKBURST INCIDENTS

At the eastern end of the Macassa orebody, mining was carried out from about the 2000 level down to the 5450 level. No large bursts were reported to accompany early mining on the eastern part of the property. Historically, rockbursts at Macassa were caused by an increasing stress on a sill pillar as the mining front came up under a mined-out level, as well as the small strain energy-type bursts that occurred during both mining and development. Serious bursting did not occur at Macassa until the 1980s.

The initial strategy to minimize bursts was to limit sill pillars to about 17 m in height, then to mine this pillar by vertical long-hole retreat. Though this allowed sill pillars to be recovered more safely, it did not prevent rockbursts. Sublevel long-hole mining was also tried with some success; however, some very large and damaging bursts, up to 3.1 M_n, were induced that required extensive stope and sill drift rehabilitation. Microseismic monitoring was initiated in the mid-1980s when a 32-geophone system was installed.

By the time the No. 3 Shaft was put into service, mining above the 6450 level had reached a mature mining stage and sill pillar bursting was common. The only large block of reserves remaining was below the 6450 level. Mining of these reserves started with benching and underhand mining below the 6450, 6600, and 6750 levels, and overhand cut-and-fill mining was initiated from the 7050, 6750, and 6600 levels. This mining, using a cemented rock fill, was designed to leave a horizontal pillar along the center of a stope. Initially, both production and ounces were the highest ever for Macassa, and there was little bursting below the 6450 level, because pillars were not yet being formed. Bursting associated with mining above the 6450 level continued to be a problem.

From 1980 to September of 1993, some 40 rockbursts greater than 1.7 M_n were reported, with 22 greater than 2.0 M_n and 4 from 3.0 to 3.1 M_n. Most of these bursts occurred above 6450 level and were located in sill or waste pillars. The majority of large bursts were not triggered by blasting and were not preceded by microseismic precursor activity. Multiple bursts were common, particularly in waste pillar areas. Pillar destressing was tried but did not become standard practice. Figure 6.2 is a typical long section of a part of the west ore zone above the 6150 level showing locations of damage from a series of rockbursts induced by overhand cut-and-fill mining.

While attention was focused on the large damaging bursts, 85% of all rockburst injuries at Macassa were caused by bursts displacing no more than 4 t of rock, and the majority of these injuries came from bursts displacing less than 1 t. About 70% of the injuries were associated with stope mining, usually as a result of scaling or drilling.

By November 1993, mining below the 6450 level had formed a center sill pillar in the 6736 stope. Development to recover this pillar by long-hole mining had begun. At the same time, mining in a small ore zone east of the Tegren Fault was being carried out by benching down the 6623 stope from the 6450 level and the 6723 stope from the 6600 level.

November 26, 1993—6723 Rockburst

On the November 26, 1993, at 8:20 a.m., a 2.5 M_n rockburst occurred at Macassa in the 6623 pillar directly above the 6600 level. This burst was followed 7 seconds later by a 2.1 M_n burst that was followed 2 seconds later by another burst of 1.7 M_n. The Eastern Canada Seismic Network recorded these bursts as a single event of 2.8 M_n.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

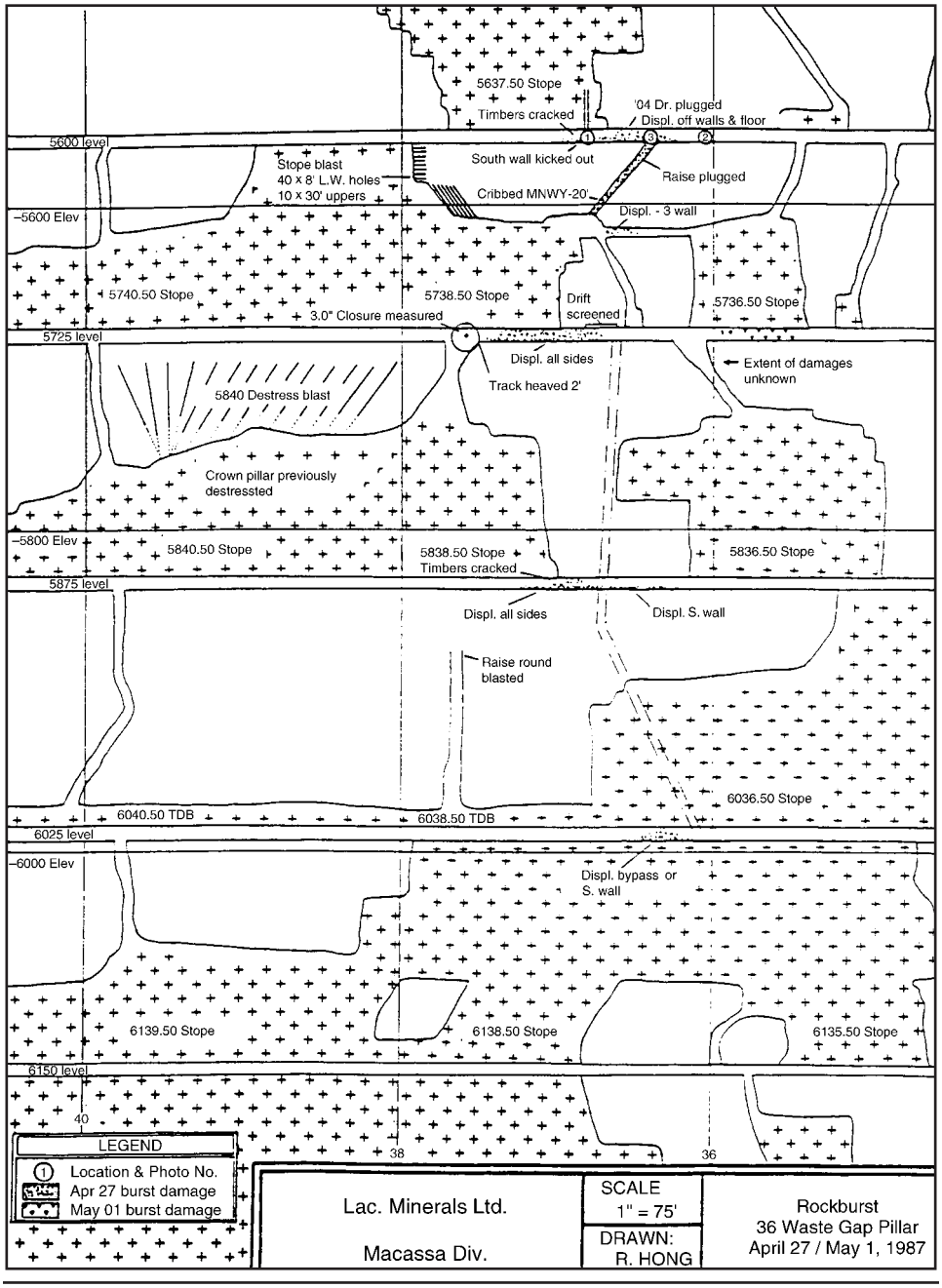


FIGURE 6.2 Longitudinal section showing location of damage caused by a series of rockbursts

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

The first burst was located in the 6623 pillar and led to massive failure of this pillar. The failed material was ejected down onto the backfill mat of the 6723 underhand stope, causing this mat to collapse and trap the two miners in the stope. The backfill mat collapsed over a distance of about 40 m where the stope width was greater than 7 m wide. The total amount of damage was about 22,000 t, of which about 14,500 t came from the 6623 pillar, 4,500 t from the hanging wall, and some 3,000 t from the backfill. The resultant cavity in the 6623 pillar extended about 43 m along strike, by 9 m wide, and arched up over 18 m above the 6600 level, as shown in Figure 6.3. Because of lack of access and the large tonnage of rock to be removed, it took 77 days to recover the bodies of the miners. Damage from the second burst caused floor heave and displaced rock along the 6450 level drift at the east end of the 6623 stope. The third burst was not picked up by the microseismic system.

It was concluded that these bursts were not triggered by the last blast in the 6623 stope, some 16 hours before the first burst, but by cumulative stress transfer effects which caused slippage along a crossover fault that ran through the pillar. Examination of ground support in accessible drifts indicated that loss of ground support functionality under static conditions was not the cause of the ground fall or the collapse of the sill mat. The ground support and cable bolts above the drift on the 6600 level, as well as the cable slings below the fill mat, were designed for static loads.

As a result of these rockbursts a new mining method was implemented that reduced exposure of the miners in the stopes. Stope access would be from well-supported drifts and raises in the footwall, with remote-controlled mucking of the ore. A paste backfill system was implemented, and the ground support installed in burst-prone ground was designed to absorb 9 kJ/sq m energy from dynamic loading.

When Macassa resumed production in June 1994, the mining method was changed to long-hole mining with footwall access where possible. Overhand cut-and-fill mining continued in some stopes on the upper levels, but all mining below the 6450 level was by longhole, except for one small underhand stope below the 7050 level. In addition to increasing ground support to contain the effects of dynamic loading, a stope reentry policy was established for long-hole blasts that included review of the microseismic activity and stope inspections prior to crews going back into stopes.

Long-hole mining did induce bursts with the larger blasts, but the combination of paste backfill, increased ground support, and the reentry policy allowed for improved rockburst control. The only serious problem caused by long-hole mining was increased dilution along narrow stopes. However, long-hole mining did result in large bursts occurring on footwall structures, which damaged scram and drawpoint drifts. In April 1997 the first production blast was taken in the 6638 long-hole stope. This blast induced a rockburst sequence that caused extensive damage throughout a large part of the mine as well as to the No. 3 Shaft.

April 12, 1997, Rockburst Sequence

A long-hole blast was initiated at 3:14 a.m. in the 6638 stope at the end of the Friday afternoon shift. Seven minutes later a 3.8 M_n rockburst occurred adjacent to the blasted panel. This was the largest burst ever recorded at Macassa; however, a magnitude for this burst was not available until the following Monday. The initial location for this burst by the microseismic system was in a waste pillar more than 150 m from its actual source location. Microseismic activity immediately migrated up-dip, spanning an area

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

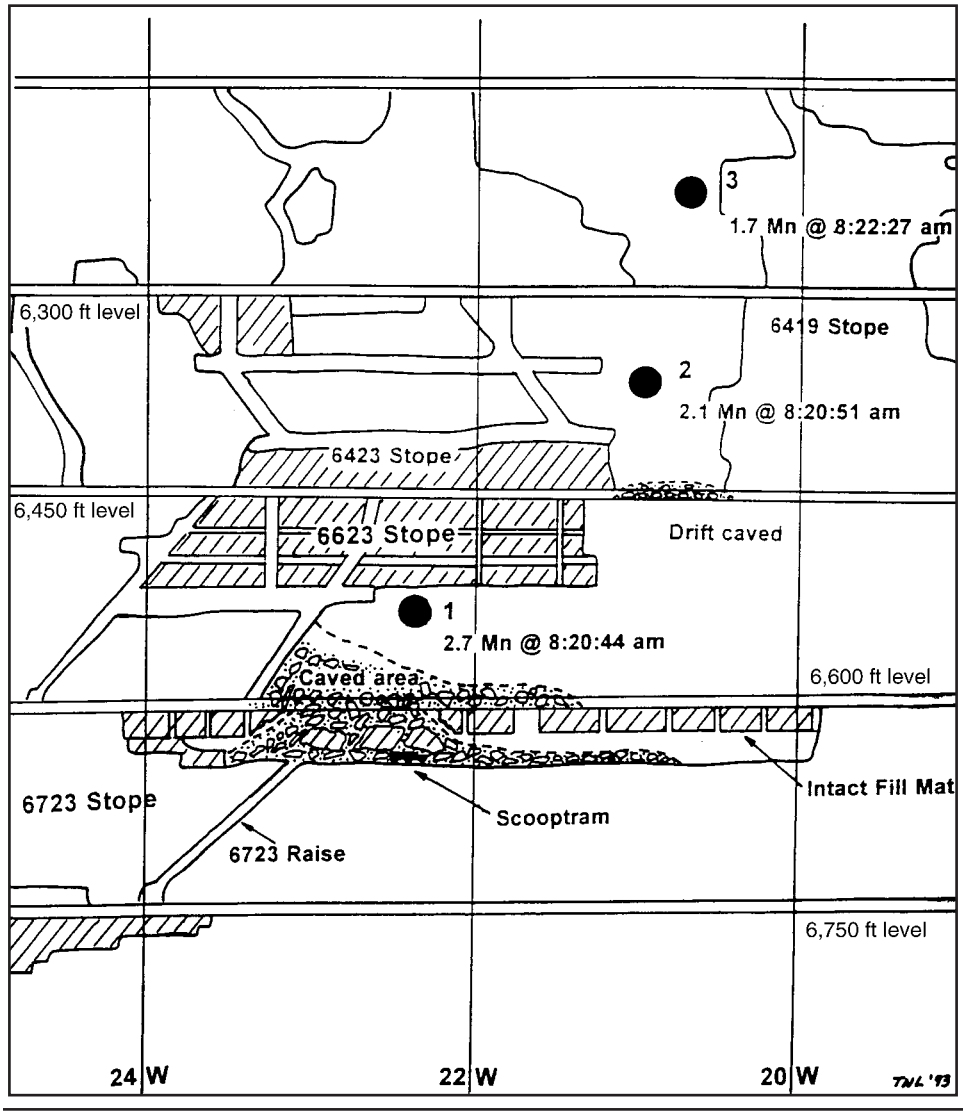


FIGURE 6.3 Detailed longitudinal section showing the November 26, 1993, rockburst locations relative to the 6723.30 stope

between the 5725 and 6750 levels and sections 30W to 41W. Included in this activity were five rockbursts ranging from 1.0 to 2.6 M_n and occurring in waste pillars below the 5725 level.

Between 4:00 a.m. and 7:26 a.m., there were no events larger than 1.0 M_n, although 313 microseismic events were recorded. Starting at 7:26 a.m. and continuing over the next hour, 14 rockbursts occurred, including large bursts of 3.7 M_n, 2.8 M_n, 2.5 M_n, and 2.2 M_n. Most of these large bursts were located by the microseismic system in waste pillars between the 5600 and 5875 levels, which had a previous history of bursting dating

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

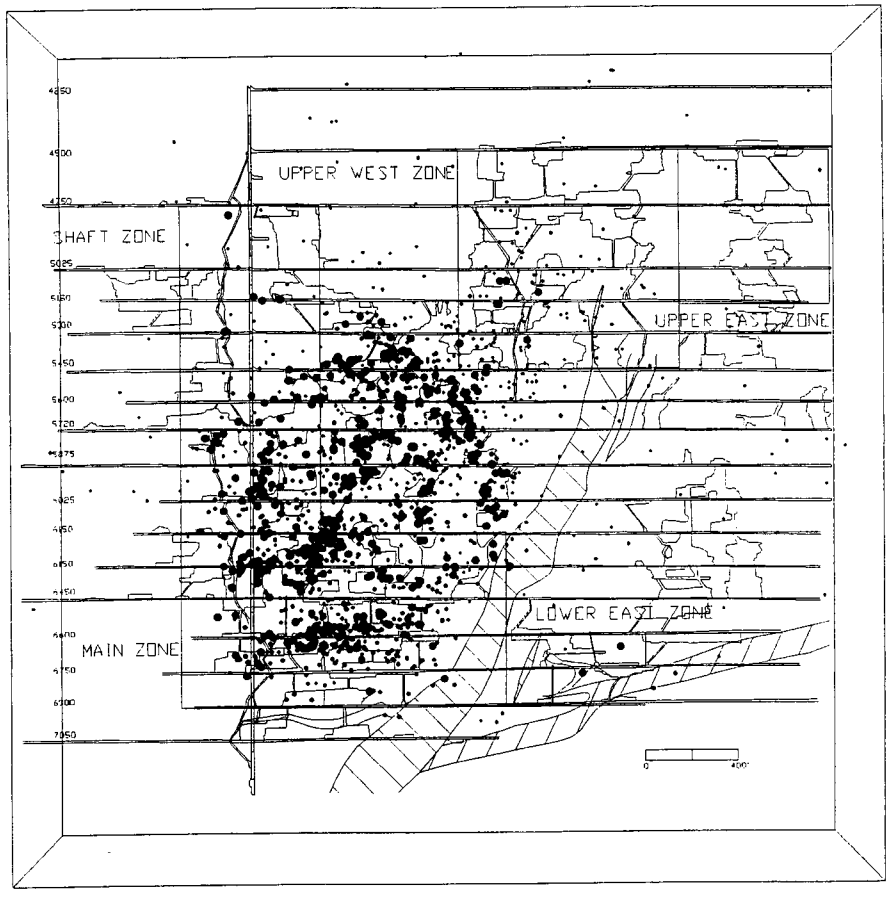


FIGURE 6.4 Seismic activity on April 12, 1997—1,716 events

back to 1987. Two smaller bursts, 1.8 M_n and 1.2 M_n, were located close to the No. 3 Shaft near the 6150 and 6450 levels. The last major burst was at 8:24 a.m. on April 12. Figure 6.4 is a plot of the seismic activity detected by the microseismic system on that day. A total of 1,716 events were recorded, and the microseismic activity, including small bursts on April 14 and 20, did not decrease to mine background, 33 events per day, until April 24, 1997.

Damage from the 3.8 M_n burst was widespread, but relatively light in the vicinity of the 6638 stope. There was minor shake-down along nearby access openings on the 6450 level, as well as along the 6450 decline into the 6638 drill horizon. Significant floor heave and stope wall closure was apparent all along the drill horizon. In the 6638 stope undercut there was significant floor heave and obvious wall closure along its entire length. The footwall side of the undercut was broken up and bagged behind the wire mesh, but the hanging-wall side was undamaged. There was heavy damage along the footwall scram drift, involving more than 500 t of displaced rock. The joint-controlled

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

damage along the scram and drawpoints was almost identical to a previous 2.6 M_n rockburst in September 1996. On the 6600 level the drift along the 04 Break under the 6638 stope was completely blocked. There were no indications of any displacement having occurred along geologic structures or contacts in the 6638 stope area. This burst did no damage to the No. 3 Shaft. It was later determined that the mechanism for this burst was a foundation failure. It should be mentioned that the blast broke well, resulting in smooth walls and brows and well-fragmented muck.

Damage from the 3.7 M_n and subsequent rockbursts was observed to be extremely heavy in a large number of accessible openings from the 5450 level down to the 6025 level between sections 32W to 42W. The 5450 and 5600 level drifts along the 04 Break were completely closed off from about section 39W to the east for an unknown distance. Major damage to the No. 3 Shaft walls and timber was observed from just above the 5875 level down to just below the 6025 level. More than 0.3 m of shaft wall closure was measured at the 6025 level station. Shaft stations and access openings out from the shaft on the 5875 and 6025 levels were heavily damaged. It appeared that at least two significant bursts occurred in the east wall of the shaft, one just below the 5875 level and one just below the 6025 level. Simultaneous with the 3.7 M_n burst, the shaft bells started ringing. This burst was located by the microseismic system about 150 m east of the shaft and just on the 5725 level. No burst damage was observed at this location; hence, it was inferred that the 3.7 M_n burst actually occurred just below the 5875 level shaft station.

Damage on levels beyond the shaft stations was mostly contained by back and wall reinforcement of bolts and chain link mesh. Below this reinforcement, particularly in basic syenite, broken wall rock was displaced into the opening. Service openings adjacent to or near the shaft, such as electrical substations or refuge chambers, suffered only minor shake-down damage. Where these openings had shotcrete applied over the bolts and chain link mesh, no damage occurred. The shaft wall reinforcement of bolts and chain link mesh prevented displaced wall rock from falling down the shaft; however, broken shaft timbers did fall. Horizontal compression effects appeared to be the predominant failure mechanism observed, and significant stope closure along the 04 Break was obvious from the 5300 to 6600 levels. Along the upper levels and in the shaft, the areas of most severe damage did not correspond with the locations of the major bursts as determined by the microseismic system.

The mine had not planned to schedule work on the Saturday following the blast in the 6638 stope. Though crews did show up for work, no work was scheduled in the 6638 stope. The microseismic system erroneously indicated no activity between 6:00 a.m. and 7:00 a.m.; hence, it was decided to send crews down at 7:00 a.m. The first cage had gone down and the second cage was being sent down when a large seismic event was felt. The crews were then all recalled to surface. Two persons were sent back underground to the 6450 level to check out the pumping system. After releasing the cage, they started out on the level when the 3.7 M_n burst occurred at 8:11 a.m. They went back to the station and noticed that shaft timber and some muck were on top of the cage. They called to surface and reported shaft damage. They were instructed to wait at the station while an action plan was formulated. When it was determined that the shaft was blocked below the 5725 level, they were instructed to go to the No. 2 Winze on the 6450 level, climb the manway up to the 5000 level, then walk back to the No. 3 Shaft to be hoisted to surface. Because of the hot and humid environment, as well as the long ladder climb, mine rescue personnel were sent down to meet them with water and provide assistance if required.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

As a result of the April 12, 1997, rockburst sequence and the problem getting the two men back to surface, the Ontario Ministry of Labour issued a stop-work order at Macassa on April 14, preventing access below the 4500 level until the seismicity settled down. This order was later amended to allow mining above the 5000 level but no shaft repair or mining below this level. The Ministry concluded that since a rockburst fatality in 1991, the mine had not demonstrated that they could manage their rockburst problem, and hence, crews could not be sent back down below the 5000 level without being endangered.

Due to limited reserves and a major expense to repair or replace the No. 3 Shaft below the 5000 level, the mine decided not to rehabilitate the shaft. Mining above the 5000 level induced very few bursts, and after about 1 year the Ministry allowed mining down to the 5300 level outside of the No. 3 Shaft pillar zone. Owing to low gold prices, high development costs, and only a few large stope blocks, the mine was closed and put on standby in the latter part of 1999.

DISCUSSION

While Macassa began mining at about the same time as all the other gold mines at Kirkland Lake, rockbursting did not become a serious problem until the 1980s. By this time Macassa was the only mine still operating, as the others had all closed by the late 1960s. The mineralization along the 04 Break was much less continuous than was the case along the Main Break. Because of this, pillar bursting at Macassa did not result in large damaging bursts until the more extensive mineralized zones in the deeper western part of the mine were mined. Incidents of small strain energy bursts did not appear to increase as the mining progressed from east to west and became deeper.

Destressing of both drift headings and pillars was more common at the east end of the mine than it was on the west end. The reason for this is not known. To deal with sill pillar bursts, the initial strategy was to limit their size, then recover them by some form of long-hole drilling and blasting. These efforts were generally not very successful, and in cases where sublevels were driven in the middle of a pillar to provide drilling access, the burst problem became more severe. Because stopes were developed on the vein, sill pillar bursts would cause the overlying timber and unconsolidated fill to collapse and result in very difficult rehabilitation to reopen sill drifts.

To minimize drift rehabilitation for sill pillar bursts, the sill pillar was moved to the center of the stope, rather than just below the level, when mining was carried out from the No. 3 Shaft. This procedure resulted in higher production benching and overhand mining, but placing the cemented rock fill was labor-intensive and time-consuming, as well as the cause of frequent cement burn injuries. Further, the time to prep and place the cemented rock fill, with at least a 1 m gap, reduced the effectiveness of this fill with respect to restricting closure.

To better deal with the rockburst problem, ground control input into mining increased. The level of ground support in stopes and access openings was increased, but this support was still designed for static loads. The microseismic system was upgraded and the number of geophones increased to 48. A surface seismograph and a macroseismic system were installed by the Canada Center for Mineral and Energy Technology (CANMET). Recovering the center pillar of the 6736 stope by long-hole mining, with

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

access from the footwall, was planned and being developed when the 6723 stope rockburst occurred.

During recovery operations, the mine was shut down, and an internal committee was formed to review bursting at Macassa and develop plans for safer and more efficient mining. It was recommended that the mine convert to long-hole mining from footwall access, increase the level of ground support to contain the effects of dynamic loading, convert to a paste backfill, and further increase ground control input into mining. All these recommendations were put into effect when mining resumed at Macassa.

Long-hole mining brought about a change in the character of the bursting at Macassa. The majority of the big bursts were now located out in the footwall, and appeared to be occurring on geologic structures. It was later determined that a number of these bursts were foundation failures occurring in a softer basic syenite rock that extended from the hanging wall out some 35 m into the footwall. This rock unit was being compressed between stiffer syenite porphyry rock units that were loaded by the high horizontal and mining-induced stresses.

The initial 3.8 M_n burst on April 12, 1997, was caused by a foundation failure. This failure essentially breached the last pillar supporting an area of some 300 m high by 150 m along strike. This resulted in immediate closure and stress transfer up-dip to the waste pillar areas below the 5725 level and initiated the rockburst sequence.

The microseismic system did not respond well during the rockburst sequence. Some time before the initial burst, there was a power interruption that caused the system to restart on Eastern time rather than on Daylight time. The source location for the initial burst only used nine geophones in the solution rather than using more of the 54 geophones that detected the event. This resulted in the wrong initial location for this event up in the waste pillar. The clock was corrected and the daily microseismic report was produced by the computer. Because the time was changed, an hour gap was created in the data plot that indicated no microseismic activity, when the mine was still seismically active. Hence, the crews were sent underground. After the three bursts at 7:26 a.m., the event buffers for the microseismic system overflowed and all subsequent waveform data was lost. The 3.7 M_n burst at 8:11 a.m. and all subsequent bursts could not be relocated.

It was not known on Saturday morning that the burst triggered by the 6636 long-hole blast was a 3.8 M_n burst, because the staff on hand could not access the computer associated with the digital seismograph. Since this was the largest burst ever at Macassa, no one should have been sent underground until a more detailed investigation of the seismicity and other on-line rock mechanics data had been carried out.

The No. 3 Shaft should not have been sited so that it intersected the 04 Break at depth. The history of losing shafts to bursting under similar conditions at Lake Shore and other mines at Kirkland Lake had been known for the preceding 60 years. This was a serious management mistake that led to premature closure of the Macassa Mine.

REFERENCES

Arjang, B., and G. Nemcsok. 1986. Review of rockburst incidents at the Macassa Mine, Kirkland Lake. *Div. Report MRL 87-21(TR)*. Ottawa: CANMET, Energy, Mines and Resources, Canada.

Canadian Rockburst Research Program 1990–1995. 1996. *Stiff Backfill*. Vol. 6. Canada: CAMIRO Mining Division. 77.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

Cook, J.F., and D. Bruce. 1983. Rockburst control through destressing—A case example. In *Rockbursts: Prediction and Control*. London: Institution of Mining and Metallurgy.

Quesnel, W.J.F., H. de Ruiter, and A. Pervik. 1989. The assessment of cemented rockfill for regional and local support in a rockburst environment, Lac Minerals, Ltd., Macassa Division. *Proceedings 4th International Symposium on Mining with Backfill*, Montreal.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 7

Falconbridge No. 5 Shaft Sudbury, Ontario

SUMMARY

The Falconbridge No. 5 Shaft Mine had a history of minor rockbursts dating back to 1955. By the mid 1980s, more than 90% of the orebody had been mined and production was concentrated in a strategic pillar, near the center of the mine. In June 1984 a series of large rockbursts occurred that, unfortunately, resulted in four fatalities and closure of the mine. For Canadian mines, this was the first clear evidence in modern times, of rockbursts being produced by slippage along prominent faults. After the mine closed, seismic activity substantially declined, until the mine was allowed to flood in 1990. This triggered more rockbursts over a 20-month period, the largest of 2.7 M_n.

GEOLOGY AND MINING METHODS

The Falconbridge Mine is located on the southeast rim of the Sudbury Basin and has been in operation since 1929. The nickel-copper tabular orebody strikes east-west for about 2,000 m on surface and extends to a depth of 1,800 m, in a triangular shape. It dips steeply to the north down to the 1200 level, then reverses and dips steeply to the south at depth. The orebody pinches and swells with an average width of 5 m. At depth the hanging wall greenstones, and especially jasperoid inclusions, are the strongest and most brittle rock, followed by the footwall norites, with the massive sulfide ore being the weakest rock.

A number of major faults intersect the orebody, which itself is on the Main Fault. This fault occurs at the contact between the breccia sulfides and norite, and is characterized by gouge material up to 1 m thick. Where there is a deviation of the contact, mineralized tangential shears break away from the fault into the hanging wall. The No. 1 Flat Fault dips at about 45° to the northeast and extends across the whole mine. Gouge on this fault varies from a few centimeters to half a meter in thickness. The Ore Pass faults also extend across the whole mine and are comprised of a faulted and sheared zone up to 15 m wide. They dip steeply and intersect the orebody at an acute angle. All these faults intersect each other on the 4025 level, just east of a hoist room pillar.

The mine is serviced by two main shafts: No. 5 Shaft extends from the surface to the 4200 level, while No. 9 Shaft extends from the 3850 level to the 6050 level, which is the deepest level in the mine. On the 3850 level No. 9 Shaft is located in the footwall norite

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

some 75 m from the orebody, but the hoist room for this shaft, for some reason, was located only 25 m from the orebody. This necessitated leaving a pillar between the 3675 and 4025 levels and extending about 200 m along strike to protect the hoist room.

Initially the upper levels were extracted with shrinkage methods, but this was soon changed to square-set mining, presumably due to poor ground conditions. This was superseded by longitudinal cut-and-fill techniques with cemented tailings. As sill pillars were formed, the orebody started to fracture, causing these pillars to be recovered using undercut-and-fill methods. By the early 1980s more than 90% of the orebody had been extracted and the major remaining ore reserves were in the hoist room pillar and a few blocks of ore directly below the No. 1 Flat Fault. It was intended that these ore reserves be recovered exclusively using undercut-and-fill methods.

EARLY ROCKBURST ACTIVITY

Between 1955 and 1983, about 600 seismic events were reported, but only 31, or 5%, caused damage of more than 1 t. Some events were felt at the town site on surface and probably had a magnitude of about 2.0, but usually no damage could be found.

An evaluation of these early bursts indicated that 54% were associated with faults and other geologic structures and the remaining 46% were associated with the mining of sill and remnant pillars. It was observed that 78% of all seismic events were below the No. 1 Flat Fault and 85% below the 2450 level.

The main rockburst control strategy, at that time, was concerned with the mining of the sill pillars, and destress blasting techniques were used on the brittle jasperoid inclusions in the hanging wall, which were presumed to be burst prone. In 1981, in preparation for mining the hoist room pillar, an 8-channel microseismic system was installed around this pillar. This system soon indicated that most of the microseismic activity was occurring in the footwall norite near prominent faults and especially the Ore Pass faults. Some minor activity occurred around development openings in the hanging wall, with very little activity in the orebody. In September 1983 the system located a large rockburst of 2.8 M_n in the footwall, about 70 m from the orebody and on the 3325 level. This burst caused severe damage to this level and the one above, near the No. 7 Shaft.

MAJOR ROCKBURST SEQUENCE

Directly below the No. 1 Flat Fault, between the 2100 to 3850 levels, a tangential fault breaks away from the Main Fault into the hanging wall, creating a wedge of ore up to 20 m wide. Some of these wide ore zones, especially adjacent to the hoist room pillar, were left in place until the 1970s. Overhand cut-and-fill stopes were then started from the 4025 and 4200 levels, directly below the No. 1 Flat Fault.

The stope above the 4025 level was completed through to the 3850 level, with generally poor ground conditions. In addition, the eastern limit of the stope was moved back about 15 m from the No. 1 Flat Fault. The stopes above the 4200 level encountered poor ground conditions, including falls of ground near the intersection of the No. 1 Ore Pass and No. 1 Flat faults. Eventually these stopes were abandoned after they had been advanced about 30 m, with the intention of recovering the remaining ore by undercut-and-fill methods from the 4025 level. The first attempt to drive an Alimak raise up to the 4025 level, for use as a boxhole for the undercut stope, was abandoned about half way up due to poor ground conditions. However, a second raise 30 m away was successful.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

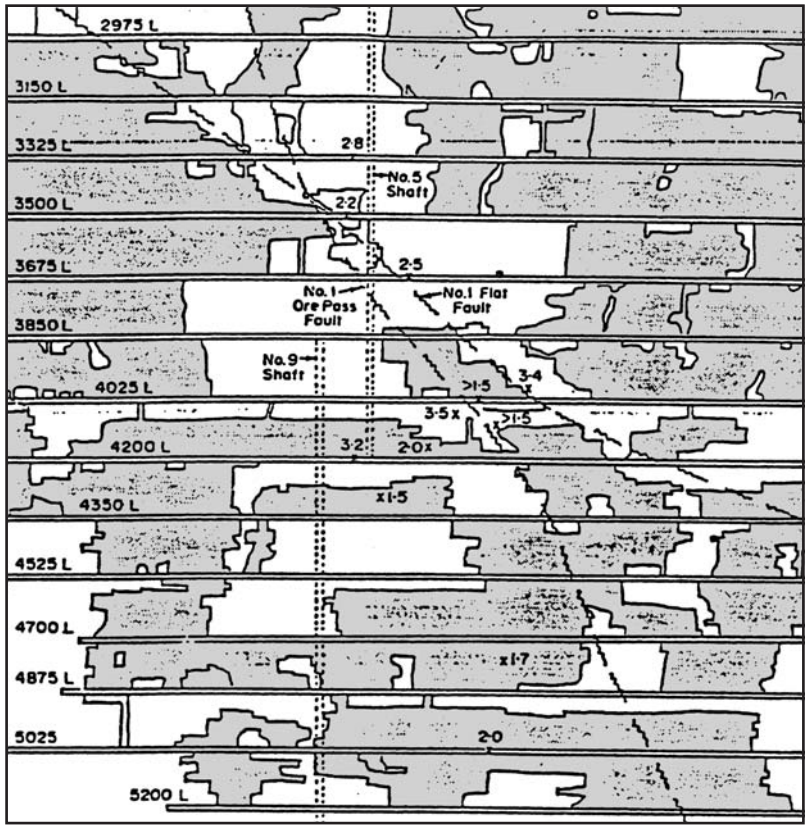


FIGURE 7.1 Longitudinal section showing location of major rockbursts

The plan to mine the wide wedge of ore with undercut methods was first to drive a 3.6 m drift along the hanging wall. Then transverse panels would be driven until they intersected either the Main Fault or the eastern end of the No. 1 Flat Fault. By June of 1984 the hanging wall drift and the first transverse panel had been mined.

On June 20, 1984, at 10:12 a.m., the first of a series of major rockbursts occurred. Over a 24-hour period, about 250 seismic events were recorded by the microseismic system. The locations of the major bursts are shown on the longitudinal section in Figure 7.1 (also shown is the 2.8 M_n event in September 1983 and two events in April 1985). Those rockbursts near the 4025 level are plotted on this level plan in Figure 7.2, which also shows the major faults.

The first rockburst, of 3.4 M_n, occurred without warning on the No. 1 Flat Fault about 8 m above the 4025 level. Unfortunately, this was about 20 m from where four miners were working in the undercut-and-fill stope. Displacements and shock waves from this rockburst caused the hanging wall and timber support to collapse inward and the overlying fill mat to rubbleize, collapse, and entrap the miners in backfill, resulting in four fatalities. Following this burst, the mine was evacuated. The next major rockburst, 3.5 M_n, occurred 2 hours later on the Ore Pass faults about 15 m below the 4025 level. This burst caused additional collapse of the backfill mat at the western end of the undercut stope and

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

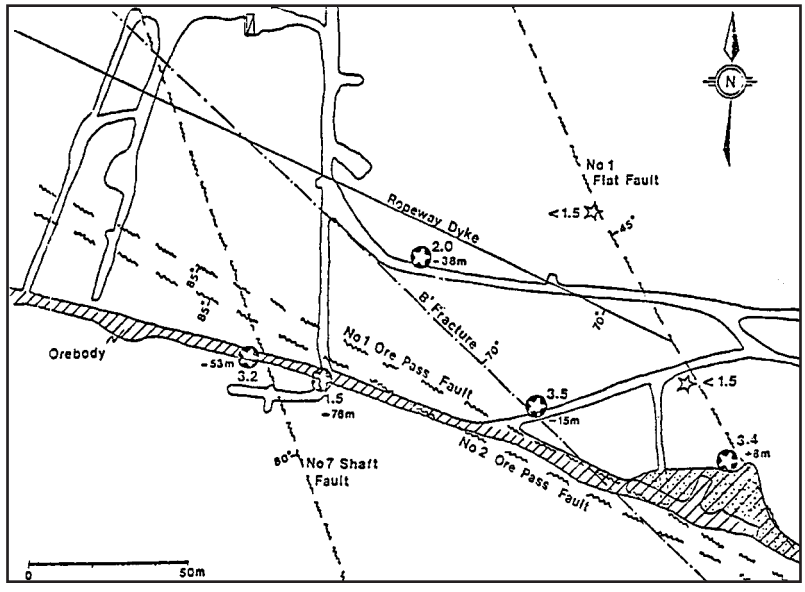


FIGURE 7.2 Plan of the 4025 level showing faulting and rockburst locations

severe damage in the crosscut above. This burst slightly injured some of the mine rescue crew trying to reach the entrapped miners, and caused this crew to be evacuated from this area until the mine settled down seismically. Eight minutes later a 3.2 M_n burst occurred on the same Ore Pass faults on the 4200 level, causing the drift in the orebody to collapse. For the rest of June 20 and 21, seismic activity was restricted mainly between the 4025 and 4350 levels and over a distance of 200 m on strike. Most of this activity appeared to be associated with the No. 1 Flat Fault, the Ore Pass faults, and the Ropeway Dyke, as shown in Figure 7.2. During June 22 and 23, seismic activity moved up-dip, and one rockburst of 2.5 M_n occurred on the 3675 level, again on the Ore Pass faults. The last of this sequence of bursts, 2.2 M_n, occurred on July 5, 1994, on the 3500 level and on the No. 1 Flat Fault.

Visual inspection of the mine workings confirmed damage between the 4025 and 4350 levels with more than 1,000 t of displaced rock, mainly within the plane of the orebody. Where the prominent faults intersected the access openings in the footwall there was generally about 1 t of shake-down rock, which unfortunately obscured the amount and direction of slippage. The exception was the rockburst on July 5, 1984, when there was no damage. From striation marks on the gouge of the No. 1 Flat Fault on the 3500 and 3650 levels, the west side of the fault had moved laterally 1–2 cm toward the orebody.

The mine was closed in July 1984 as a direct result of the rockburst incidents and limited ore reserves. Seismic activity continued sporadically with magnitudes up to 2.2. In November 1990 the mine was allowed to flood, and in May 1991 significant seismic activity again began to occur in the mine. Over a 20-month period, 22 seismic events greater than 0.1 M_n were recorded, with four of them greater than 2.0 M_n, the largest being 2.7 M_n on April 25, 1992. It is estimated that in May 1991 the floodwater would have reached the 4800 level, in April 1992 the 2800 level, and in December 1992 it is known that the flooding had reached the 1200 level. It would appear that flooding reactivated slippage on the prominent faults.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

DISCUSSION

The first 3.4 M_n rockburst on the No. 1 Flat Fault was probably caused by mining in the adjacent undercut-and-fill stope. As there was no warning, the triggering mechanism was probably a reduction in the clamping stress, rather than an increase in the shear stress. Normally, a gouge-filled fault would slip gradually and nonviolently. However, in the vicinity of the 4025 level there was a bulge in the No. 1 Flat Fault creating an asperity, which would have to be sheared through.

Once movement occurred on the No. 1 Flat Fault, it allowed other faults, noticeably the Ore Pass faults, to become active. Almost all of the seismic activity and all of the damage were confined to a wedge of rock, bounded by the No. 1 Flat Fault and Main faults, and extending about 200 m into the footwall norite. There was no microseismic activity east of the No. 1 Flat Fault and very little activity around development openings in the immediate hanging wall. It should be pointed out that while only seven geophones were initially installed with the microseismic system, data from this system allowed for very accurate source locations of the seismic events. Identification of individual rockburst during the sequence was made based on a time correlation with the same bursts recorded on the surface seismograph that the mine had installed at No. 5 Shaft.

As illustrated in Figure 7.2, the Main Fault, No. 1 Flat Fault, and Ore Pass faults intersect each other in the same area. One of the strategies developed in Ontario mines in the 1940s was to mine against major faults as soon as possible, then to retreat away from them. In this case, more than 90% of the orebody had already been mined. Numerical modeling studies indicate that retreating away from a fault results in a succession of small slippages on the fault and relatively low releases of seismic energy; whereas, advancing toward a fault is characterized by a major seismic event when the stope intersects the fault.

An indirect contribution to the rockbursts at Falconbridge No. 5 Shaft was the decision to place the hoist room for No. 9 Shaft next to the orebody. This meant a pillar had to be left to protect the hoist room. Initially this pillar did provide protection, but by the early 1980s it was heavily stressed, cracks were appearing around the edge, and both the shaft and hoist motor were indicating some movement. If the hoist room had been placed on the other side of the shaft, then probably a pillar would not have been required. Mining in this central area of the mine could have taken place earlier, at a time of less overall extraction and under a lower stress regime.

REFERENCES

Davidge, G.R. 1984. Microseismic monitoring at Falconbridge Mine, Falconbridge, Ontario. *CIM Bull*. 77(868):45–49.

Hedley, D.G.F., S. Bharti, D. West, and W. Blake. 1985. Fault-slip rockbursts at Falconbridge Mine. *Proceedings 4th Conference Acoustic Emission/Microseismic Activity,* Pennsylvania State University. Clausthol, Germany: Trans Tech Publications.

Ng, L.K.W., G. Swan, and M. Board. 1993. The application of an energy approach in fault models for support design. *3rd International Symposium Rockbursts and Seismicity in Mines*, Kingston, Ontario. 387–391.

West, D. 1985. A case history of rockbursts at Falconbridge Mine. *CIM Seminar*, Sudbury, Ontario.

Wetmiller, R.J., C.A. Galley, and M. Plouffe. 1993. Post-closure seismicity at a hard-rock mine. *3rd International Symposium Rockburst and Seismicity in Mines*, Kingston, Ontario. 445–448.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 8

Strathcona Mine Onaping, Ontario

SUMMARY

The Strathcona Mine did not have a history of rockbursts when mining of the regional sill pillar began in 1984. More than 80% of the orebody had been mined, and it was expected that this sill could be recovered with minimal bursting problems. In June of 1988 a series of rockbursts occurred in the central part of the sill that eventually led to the end of production in the Main Ore Zone. Mining in the Deep Copper Zone at Strathcona also induced rockbursts starting in 1999 as the first sill pillars were formed. However, the mine has been able to deal with the rockburst problem and continue mining as a result of increased ground support and close cooperation between mining and ground control.

GEOLOGY AND MINING METHODS

The Strathcona Mine lies on the northern rim of the Sudbury Basin. The Main Ore Zone at Strathcona has a strike length of about 800 m and varies in thickness from 10 m to 200 m. This massive sulfide-nickel orebody has a northeast strike and dips from 45° to 80° to the southeast. The ore predominantly occurs in a breccia unit referred to as the footwall breccia. The hanging wall is a mafic norite unit that grades upwards to a felsic unit. Below the footwall, breccia is a granite gneiss complex of rocks. In the central part of the ore zone, these geologic units are crossed by an olivine diabase dyke, which strikes west to northwest and dips moderately to the north. The ore is very weak when compared to the wall rocks.

Mining in the Main Ore Zone has been carried out from the 1875 level down to the 2900 level, 570 m to 900 m below surface. Multilevel mining began in 1968 using overhand post-pillar cut-and-fill, but by the mid-1970s an increasing number of stopes were being mined by blasthole methods. Annual production was about 2.1 million metric tons. By 1980 a long sill pillar in the middle of the orebody was forming, and plans for extracting this central pillar changed from cut-and-fill mining to blasthole, utilizing transverse panels. Figure 8.1 is a longitudinal section of the central part of the mine showing the irregular sill pillar, as well as the main dyke and a small offshoot dyke. The large central part of the sill pillar is somewhat similar to the No. 9 hoist room pillar at the Falconbridge Mine.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

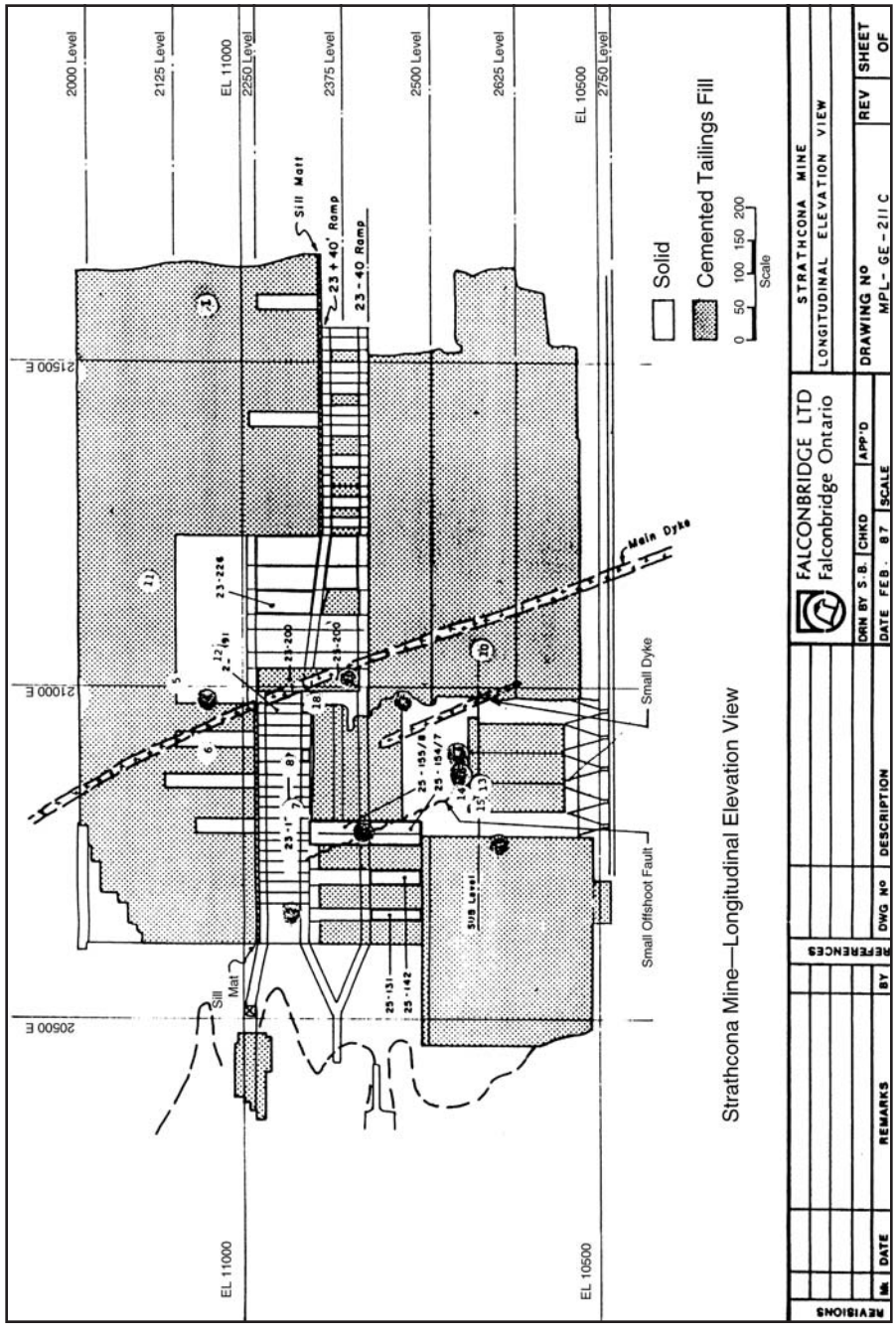


FIGURE 8.1 Main sill pillar area—Strathcona Mine

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

By late 1984, about 80% of reserves had been mined, and rockburst incidences during sill pillar mining increased dramatically. The average annual reported bursts during the period from 1971 to 1983 was 7, whereas 74 reported bursts occurred in 1984. By early 1985 it was decided to suspend sill pillar mining in the central part of the mine until a microseismic monitoring system was installed.

The Deep Copper Zone at Strathcona is located some 300 m in the footwall of the Main Ore Zone and is a complex network of irregular massive sulfide veins up to 6 m in width. The overall strike of the veins is east-west over a distance of about 350 m, and they dip from 30° to 40° to the south. These veins are soft compared to the hard felsic gneiss wall rock and consist predominantly of chalcocite, cubanite, and pentlandite, with an overall ore grade of almost 7% copper. This ore zone is serviced by the No. 1 Shaft, which was deepened in the 1980s down to the 5200 level (1,585 m below surface). Production began in 1988 on the 4200, 3900, and 3700 levels and currently is about 450,000 t per year using narrow-vein, mechanized, overhand cut-and-fill mining.

During early mining and up to the formation of 30 m sill pillars, ground conditions were good and little seismic activity accompanied mining. However, when the 4200 level stopes were advancing up to the 3900 level sill in early 1999, seismic activity and rockbursts were induced, similar to bursting encountered in other deep narrow-vein mines in Ontario.

ROCKBURSTS IN THE MAIN SILL

The initial concern in mining the 500 m long sill pillar was the central part, where an 80-m-long by 85-m-high section of the sill was intersected by the olivine-diabase dyke. The first panel to be mined was 25-200, which crossed this dyke (Figure 8.1). When the undercut for this panel intersected this dyke a strain energy burst occurred. Mining of this panel, as well as the 25-154 panel, beginning in October 1985, resulted in bursts of M_n 2.2, 2.7, and 3.2 during November and December. Damage from these bursts was in the footwall of the orebody along the access ramps from the 2300 and 2500 levels, opposite the 25-154 stope. The damage appeared to coincide with structures that were conjugate to the dyke. There was essentially no seismic response along the dyke or out in the hanging wall.

To control damage along footwall access drifts to the main sill, cable lacing was added to the standard ground support of bolts and screen in critical areas. This was the first usage of South African type lacing in North America. Mining the 23-226 panel began in April 1986. Blasting in this stope induced a large seismic response, culminating in an M_n 3.0 burst following a production blast on August 2. Both the access ramp and the 23-226 overcut were damaged. The damage along the access ramp was contained by the ground support with cable lacing, but the overcut was not laced and so severely damaged that mining was stopped and the stope filled. During 1986 some 32 seismic events greater than M_n 1.0 accompanied mining along the main sill. Mining in the sill continued in 1987 with reduced seismicity, as only 16 events greater than M_n 1.0 were induced. The checkerboard-panel-extraction–type sequence was allowing mining to progress and appeared to be minimizing bursting.

By early 1988 production from the main sill had dropped by a factor of three, and seismicity was also decreasing, as only five events greater than M_n 1.0 were reported

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

during the first 5 months of 1988. In June 1988, however, production mining in 25-143 stope and driving the 23-233 overcut led to a rockburst sequence over a 90-hour period that included 12 bursts greater than M_n 1.0, with 8 of these bursts larger than M_n 2.0, and 3 of M_n 2.7, as indicated in Figure 8.1.

The 23-233 overcut was blasted at 10:52 p.m. on June 17, generating very little seismic or microseismic activity. An M_n 1.8 burst occurred with the production blast in the 25-143 stope, followed by a large microseismic response, and an M_n 2.2 burst on June 18. Seismic and microseismic activity had appeared to settle down, when at 5:28 a.m. on June 19 an M_n 2.2 event occurred near the 23-233 overcut, followed by almost continuous microseismic activity. A 2.7 M_n burst occurred in the upper west corner of the large central sill some 25 minutes later, and within the next 50 minutes the continuing seismic response included bursts of M_n 2.4 and 1.8, also up in the large central sill. At 1:30 p.m. on the 19th the seismic activity shifted down to the pillar below the central sill, with an M_n 2.2 event at 8:40 p.m., followed by events of M_n 1.0 and 2.7 during the next 4 hours. The seismic response then shifted back up to the main sill where a 2.2 M_n burst at 2:53 p.m. on the 21st ended the sequence. These bursts were all located along structures in the footwall. Damage from this burst sequence was mostly contained where there was cable lacing along the back and walls of access drifts, but rehabilitation was required.

As a result of this bursting, mining was stopped in the central sill area pending a review of the stability of the central sill and reevaluation of the mining plan. In 1990 pillar mining was resumed in the 25-39-D4 stope located in the eastern part of the main sill. This stope was mined almost to completion when an M_n 2.6 strain burst occurred at the entrance to the stope. An attempt to destress was ineffective, and management decided to stop all production mining along the main sill. Production mining along the main sill was deferred pending further review. It was never resumed.

ROCKBURSTS IN DEEP COPPER

In June of 1999 a 3.0 M_n fault-slip type burst occurred as the 42 footwall stopes were mining up to the 39 sill pillar. During the next year seismic activity and bursting increased significantly as mining continued to advance up to the 3900 and 3700 levels. Some 25 bursts were reported with 5 greater than M_n 2.0. The largest two were M_n 2.6. Most of the bursts were located on structures in high-stress zones along the footwall, primarily in narrow stope sections.

To minimize damage from bursting, ground support in the back and walls of narrow stopes changed to include a combination of grouted rebars, mechanical bolts, #9 welded wire mesh, and 12-foot-long Superswellex. Standard reinforcement of slot entries was increased utilizing strand-loc and fiber-reinforced shotcrete where necessary. In wide sections along stopes, shotcrete pillars are used for temporary support. Stope mining is very closely monitored using a microseismic system, and wall destressing is carried out when required.

DISCUSSION

The conversion from cut-and-fill to blasthole stoping at Strathcona led to the formation of large pillars in the central part of the main sill. It was initially presumed that the soft ore would yield as the sill was mined, hence, minimizing bursting. This was generally

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

the case along the east and west ends of the sill, but not in the central part where the large pillars were left.

Initial mining in the central part of the sill indicated that bursting along structures in the footwall could be expected, much as was the case during mining the hoist room pillar at Falconbridge No. 5 Shaft. After the first strain energy burst along the dyke, the majority of seismic events occurred on structures conjugate to this dyke or along small offshoot dykes. After the bursts in late 1985, the strategy of cable lacing and checkerboard type panel extraction appeared to be working well, and it was presumed that the majority of the sill could be mined. Mining the sill at a lower extraction rate also seemed to be working, as burst incidences were also reduced, and it was presumed that the bursting was having a destressing effect on the central part of the sill. Hence, the burst sequence that was initiated in June of 1988 was not expected.

In hindsight, it is apparent that the mining to the west of and in the central sill was causing this sill to approach an unstable equilibrium state, similar to that at Falconbridge No. 5 Shaft. The displacement adjustments along structures in the footwall owing to bursting and closure had apparently reached the point where just a small amount of mining would result in destabilizing the entire central sill pillar. It is not clear that the initial bursting associated with production blasting in the 25-143 stope started the burst sequence, as the microseismic response did not indicate stress or displacement effects being transferred to the central or lower sills. Further, the face advance round in the 23-233 overcut did not have a seismic or microseismic response that might also indicate the impending instability of the central sill. Hence, the uncontrolled bursting following the M_n 2.2 burst at 5:28 a.m. on June 19th was unexpected, based on the microseismic activity prior to that day.

As was the case at Falconbridge No. 5 Shaft, an uncontrolled burst sequence can occur without a well-defined microseismic buildup. It should be remembered that this was also the case with the burst sequences in remnant pillars at both Quirke and Campbell, where bursting occurred in old, mined-out areas and was not associated with present-day mining. There were no indications that a large area of either mine would suddenly become unstable and that this instability would continue for 2 or more days.

Following the burst sequence at Strathcona, Falconbridge spent a great deal of time and effort attempting to understand what happened utilizing geological, mining, rock mechanics, and observational data. In addition, seismic velocity surveys were carried out through the central sill pillar, and the mining and bursting sequences were studied in numerical models. The results of all these studies and analyses indicated that the central part of the sill was likely destressed and that a planned mining sequence could resume. However, by the time of completion of this work, the price of nickel had declined to such a degree that resuming mining along the main sill was not economic.

Bursting during sill pillar mining at Deep Copper was expected. Stiff cemented rock fill was used for the first few cuts above the 3900 and 3700 levels to form a strong and relatively stiff backfill mat to facilitate eventual recovery of the underlying sill pillars. It was felt that this strategy would allow for a higher recovery of sill pillars as they were mined. Drift and fill mining of sills is also planned to reduce spans, and long-hole uppers will be used where possible. To control burst damage, the very resistant ground support now being installed will be supplemented by heavy gage screen strapping between bolts along the footwall. In addition, more detailed microseismic monitoring and local destressing will be used to minimize bursting and its affects.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

REFERENCES

Davidge, G.R., T.A. Martin, and C.M. Steed. 1988. Lacing support trial at Strathcona Mine. *Proceedings 2nd International Symposium Rockbursts and Seismicity in Mines*, Minneapolis, MN, June 1988. Rotterdam: A.A. Balkema.

MacIssac, H.S., and G. Swan. 2000. Case Study: Strathcona Deep Copper Mine. In *Underground Mining Methods Handbook, 3rd ed.* Edited by W. Hustrulid. Littleton, CO Society for Mining, Metallurgy, and Exploration, Inc.

Swan, G. 1992. Falconbridge's Strathcona Mine. *CANMET Special Report SP92-1E, Rockburst Handbook for Ontario Mines*. Ottawa: CANMET, Energy, Mines and Resources, Canada. 285–296.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 9

Creighton Mine Sudbury, Ontario

SUMMARY

The Creighton Mine is now the deepest operating mine in North America and the most burst-prone mine in Canada. Since the first reported rockburst in 1934, mining methods, stope sequencing, and ground support have all been modified to deal with increasing incidents of bursting with depth. Sill pillar bursting has been dealt with by a number of mine design changes, the latest being conversion to a pillarless top-down, center-out mining geometry. Routine destressing of all development openings below the 6000 level is carried out, and all openings are reinforced to contain the effects of nearby rockbursts.

Formal rock mechanics input for dealing with rockbursts began in the early 1950s and has continued to date. The outstanding safety record at Creighton, regarding the rockburst hazard, attests to the success of this program.

GEOLOGY AND MINING METHODS

The Creighton Mine is located on the southern rim of the Sudbury Basin. The massive sulfide-nickel orebody extends from the surface down to below the 7530 level, 2,287 m deep (Figure 9.1). The orebody is irregular in both size and shape, with an overall dip of about 65° to the northwest. From the 6600 level down to the 7200 level the shape of the orebody was somewhat equidimensional, but on the 7400 level it has become more elongate, 260 m × 60 m, striking north-northeast. The footwall rocks are granites and gabbros, while the hanging wall is primarily norite. Below 6600 level the hanging wall of the orebody is increasingly granites and gabbros as the ore departs from the norite sublayer and intrudes the footwall. The orebody is intersected by lamprophyre dykes, as well as shear zones, that generally strike northeasterly, and occur in both the hanging and footwall. A north-south striking quartz diorite dyke extends into the footwall from the norite sublayer and is present from the surface to below the 7200 level.

Mining operations began in 1901 from an open pit on surface. Since that time, many different mining methods have been utilized, including shrinkage, blasthole, panel caving, square-set, overhand and underhand cut-and-fill, and vertical retreat blasthole. As mining has gone deeper, ground control input has become more important in mine design, as well as determining development and stoping sequences. The high horizontal

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

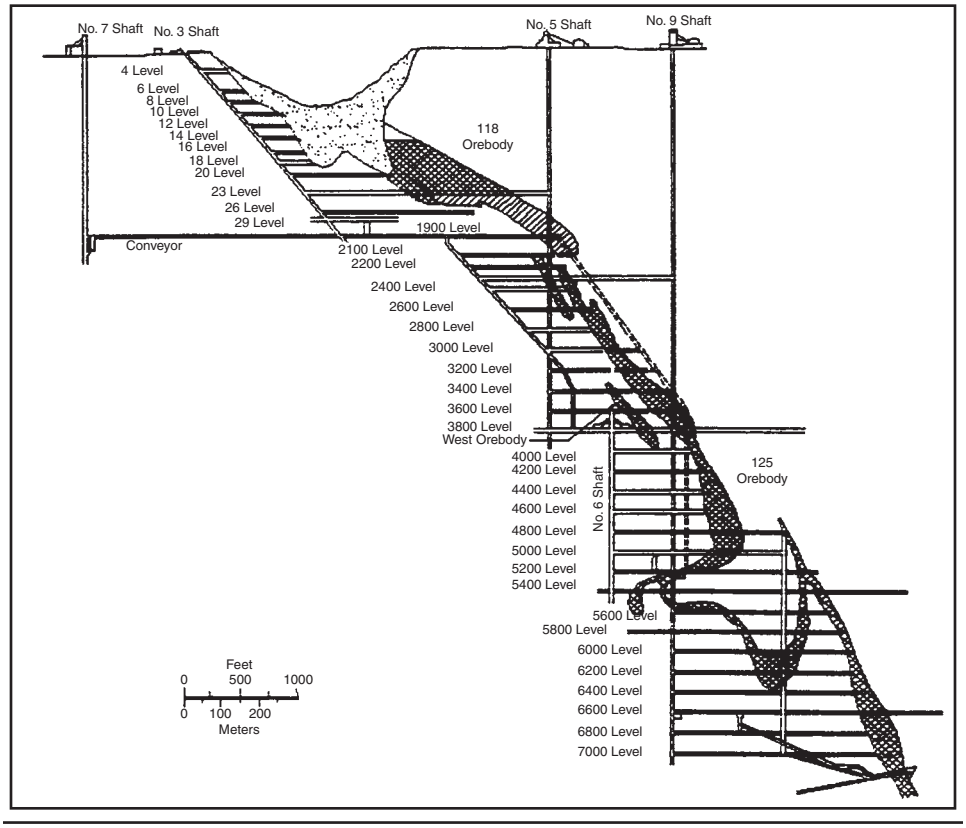


FIGURE 9.1 Generalized cross section—Creighton Mine

stress environment below the 7000 level has led to a pillarless mining sequence where no sill or crown pillars are created.

Top-sill development is now carried out in the low-stress environment in or just below the backfill of the overlying stope. Because the bottom-sill development is in virgin ground, high-stress conditions must be taken into account. Level development is sequenced in an inverted-V pattern to push stress to the abutments, while destressing is carried out to minimize the risk of strain bursting. Top-down and center-out mining blocks are also dimensioned and sequenced to start in a low-stress environment and transfer stress to the abutments.

Experience and numerical modeling are used to establish block dimensions, as well as yielding pillars. Microseismic systems are used to monitor and track seismicity, and, from this data, procedures have been developed to assess the seismic risk. Rockburst incidents associated with day-to-day mining have been a serious operational problem since 1984.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

ROCKBURST INCIDENTS

The first recorded burst at Creighton occurred in 1934 on the 2300 level, at a depth of 700 m. Below this depth, bursts have been localized primarily in crown and remnant pillars. Bursts in isolated development openings began to occur below a depth of 1,200 m. And, at a depth of 2,000 m, bursts started to occur with sill development at the beginning of production from a level.

Early reports described "induced" bursts associated with pillars after about 70% of a mining block was extracted. "Inherent" bursts were reported to be occurring in development openings. In addition, near the end of a mining block, unlocated induced bursts were reported that were presumed to occur out in the walls on the periphery of pressure domes. Since 1934 the mine has investigated more than 1,300 rockburst incidents.

Most rockbursts are related to day-to-day mining and are the result of creating pillar geometries, whereas most strain bursts have been associated with geological structures. At Creighton a 1.5 M_n burst occurs about once every week, a 2.0 M_n burst about once a month, and a 3.0 M_n burst about once a year. In 1980 a 16-geophone microseismic system was installed to monitor increasing seismic activity with depth. This was the first microseismic system installed in Ontario. The majority of rockbursts since 1984 have all occurred in the lower levels around the active mining zones.

On July 6, 1984, a rockburst sequence followed a 4.0 M_n event that occurred about 150 m out in the hanging wall of the 125 orebody between the 3200 and 3400 levels. The subsequent rockbursts (five events of 2.1 to 2.5 M_n) occurred in and along footwall development drifts between the 3200 and 3800 levels, and between the No. 5 Shaft and the orebody. These bursts were more than 200 m from the initial large burst. It was postulated that the 4.0 M_n event occurred on the western edge of the existing hanging wall subsidence zone and was triggered by hydraulic lime injected into this zone just prior to the summer shutdown. This event changed the stress regime in this area and led to the rockburst sequence. As a result of this bursting, the International Nickel Company of Canada, Ltd. (INCO) initiated a major in-house research and operations program to deal with rockbursts.

On May 25, 1998, at 11:47 a.m., a rockburst of 3.9 M_n occurred in the hanging wall of the 400 orebody at about the 7300 level, as shown in Figure 9.2. This burst was followed by numerous smaller events up to 2.1 M_n, which migrated upward and into the footwall, leading up to a 3.5 M_n burst about 12 hours later. The initial burst was located about 150 m in the hanging wall between the 7200 and 7300 levels. Subsequent seismic events were mostly located in the hanging wall and footwall outside of the mine workings. The 3.5 M_n burst occurred in the footwall at about the 6800 level, near the fresh air raises from 6800 to 7200 levels. The most likely mechanism for the 3.9 M_n burst was slippage along a hanging wall structure, identified later by cumulative plots of microseismic data. It was postulated that slippage occurred due to unclamping of the structure from the regional stress because of ore extraction above the 7200 level. This changed the stress regime in the area, resulting in displacement adjustments and aftershock seismic activity, including the 3.5 M_n event. It was also postulated that this second event was due to movement on a shear zone in the footwall. This event was also followed by seismic activity during the next 12 hours. Damage from the initial event was widespread shake-down and cracking of shotcrete in backfilled areas. Damage from the second event was also widespread shake-down, but this event was followed by small local events that resulted in more typical burst damage. Total damage from both bursts and aftershocks was in the 500 t range, but most of this damage was contained by the reinforcement

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

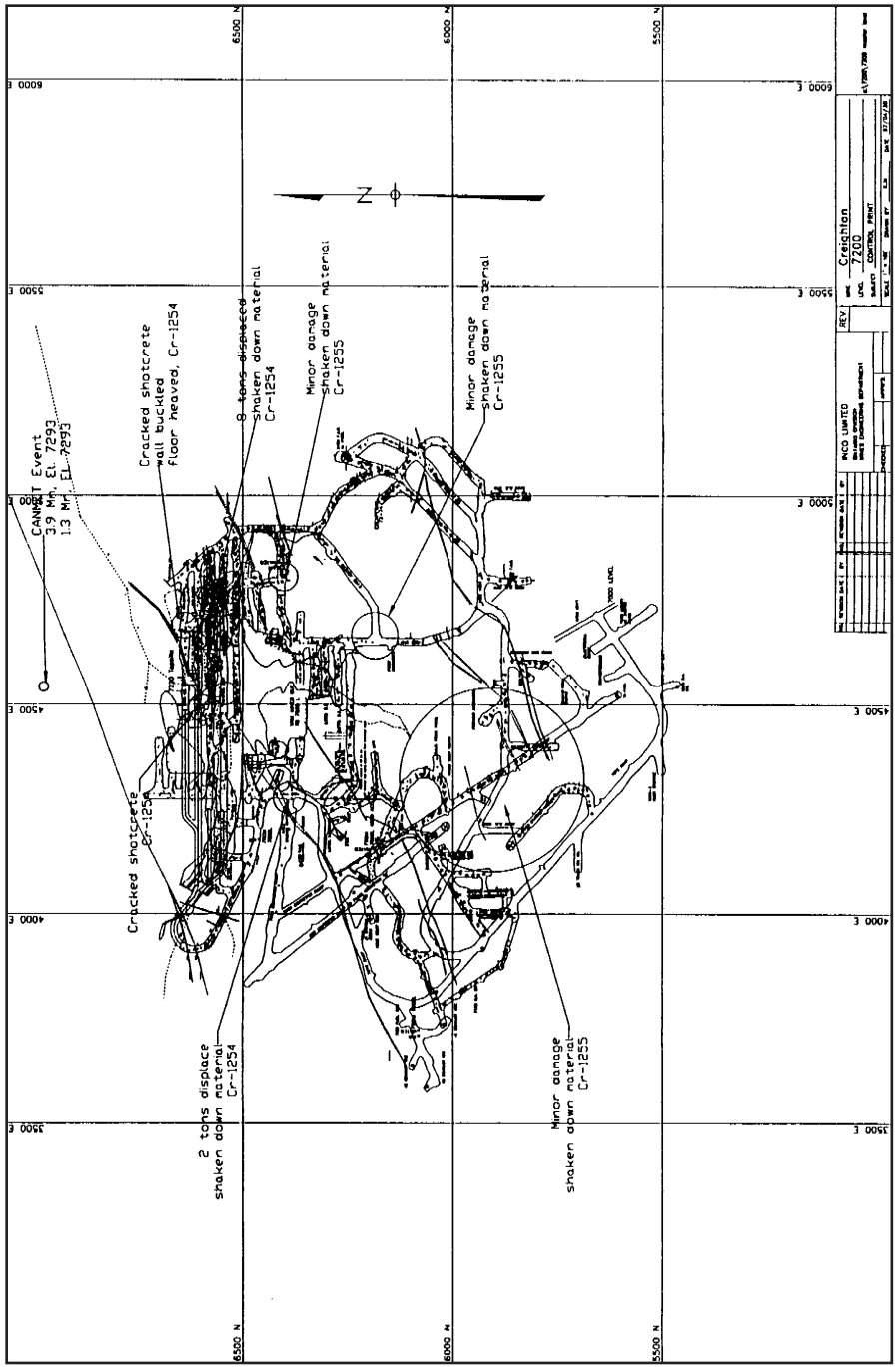


FIGURE 9.2 Plan view of 7200 level—Creighton Mine

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

except in two areas. One of these areas was along an abandoned drift on the 6900 level and the other was at a 25-year-old intersection reinforced with conventional rock bolts and screen. Damage from these large bursts was unusually light.

DISCUSSION

In addition to now being the deepest operating mine in North America, Creighton has dealt with a rockburst problem longer than any other mine discussed in these case histories. Further, despite being the most burst-prone Canadian mine since 1984, the safety record for Creighton is unmatched when compared to other mines with a similar burst history. While this may be partly due to the nature of the deposit, massive sulfide as opposed to narrow vein, it is also due to the long history of rock mechanics input into mine design and stope sequencing.

Measures to control rockbursts at Creighton have been directed by a central rock mechanics group since the 1950s. Mine layouts that resulted in yielding pillars were developed, as well as routine destressing of development drifts below an 1,800 m depth. All shafts were placed well into the footwall, their sections changed from rectangular to circular, with concrete lining, and shaft destressing was used as required for advancing through bursting ground.

When bursting initially became a problem, square-set mining with hydraulic sand was used to deal with it. When major rockbursts started to occur in sill pillars, the mining method was changed to a center lead stope geometry with yielding rib pillars. Use of cemented backfill in the 1960s allowed the mining method to be changed to a mechanized overhand cut-and-fill. To eliminate captive equipment, a retreat mining sequence was developed that progressed from hanging to footwall. Rib pillars were mined later by underhand cut-and-fill. Sill pillar bursts again became a problem, particularly along narrow sections of the orebody, as stopes were advanced to within about 20 m of the level above. The mining front was then changed back to a center lead stope geometry by creating a central destress slot across the orebody using vertical retreat methods. This lead to primary vertical retreat mining down to the 7200 level. Below this level the present top-down, center-out mining method was established.

Ground support used at Creighton has also changed with time in order to contain the effects of nearby rockbursts. Timber or steel sets with lagging have given way to steel mesh with mechanical bolts, which have now been replaced by yielding bolts, cables, screen, and shotcrete. The ground support used at present has proved to be very effective in containing rockburst damage.

The original seismic monitoring system at Creighton has been expanded and improved to include a number of different whole-waveform microseismic and seismic systems. In addition to day-to-day seismic monitoring of mining, research into all aspects of mining seismicity has been carried out since the mid-1980s. Results of this research have led to a much better understanding of the wall rock response to mining at Creighton.

Stope reentry following a production blast is based on a risk assessment derived from microseismic decay times following different-sized blasts in different zones of the orebody. In addition, an expected burst magnitude, the probability of its occurrence, and its likely source locations are also determined. These parameters are all derived from historical seismic data, reflecting mining experience. However, none of these data indicates that a very large seismic event may be ready to occur out in the walls, as has been the case at Creighton. These large events are due to the overall mine geometry and

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

resulting wall rock deformation or closure. In addition, they do not occur at regular intervals and generally are not preceded by any seismic precursors. Hence, their occurrence is almost always unexpected.

REFERENCES

Dickout, M.H. 1957. Rockburst control at the Creighton Mine of the International Nickel Company of Canada Limited. *Proceedings 6th Commonwealth Mining and Metallurgical Congr*ess, Canada. 385–388.

Oliver, P.H., T. Wiles, P. MacDonald, and D. O'Donnell. 1987. Rockburst control measures at Inco's Creighton Mine. *Proceedings 6th Conference on Ground Control in Mining*, West Virginia.

Morrison, D.M., and P. MacDonald. 1990. Rockbursts at Inco Mines. *Proceedings 2nd International Symposium Rockbursts and Seismicity in Mines*, Minneapolis, Rotterdam: A.A. Balkema. 263–267.

Thibodeau, D. 1999. *Deep Mining Considerations at Creighton Mine.* MASHA Ground Control Bulletin, Issue No. 5, October, 4 pp.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 10

Copper Cliff North Mine Sudbury, Ontario

SUMMARY

Rockbursting became a serious operational problem at Copper Cliff North in the mid-1980s as a result of pillars formed by the checkerboard-panel mining layout. After the mining geometry was changed to center-out paneling, the rockburst problem was brought under control.

GEOLOGY AND MINING METHODS

The Copper Cliff North Mine is located on an offset from the main nickel irruptive. The nickel-copper disseminated to massive sulfide mineralization occurs along a strike length of more than 4,000 m and extends from surface down to about a 1,650 m depth. The largest ore zone is the 120 orebody, which varies along strike from about N20°W to N65°W with a near vertical dip, is about 350 m long, and varies in width from about 7.5 m to 24.5 m. The dip of this orebody is overturned below the 2200 level, so that it dips about 75° toward the shaft. This has caused the major access drifts to be located in the hanging wall instead of the footwall. The host rocks are primarily metasediments with the hanging wall more silicified than the footwall. The direction of major principal stress is about normal to the orebody.

Production began in 1896 and continued until production was halted in 1978. Initial mining was by shrinkage stoping, which was later changed to blasthole mining methods. The mine was reopened in 1983, with vertical retreat mining (VRM) carried out in several different ore zones at a production rate of about 2,500 t per day. The primary VRM stope panels in the 120 orebody were originally designed to be 12 m along strike with 24 m pillars between stopes. This design was based on a concern regarding the stability and mode of failure of the pillars. The primary stopes were to be filled with cemented rock fill and the secondary stopes with only rock fill. Rock fill was delivered to stopes through a system of fill raises from surface. Figure 10.1 is a longitudinal section showing the central part of the mine.

A checkerboard pattern of stopes was mined by drilling 165-mm-diameter holes down from the 3600 level to the 3935 level. Initial success with these stopes indicated that intermediate sublevels were not necessary and that panel height could be increased. Hence, mining above would be carried out from the 3200, 2800, and 2400 levels using

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

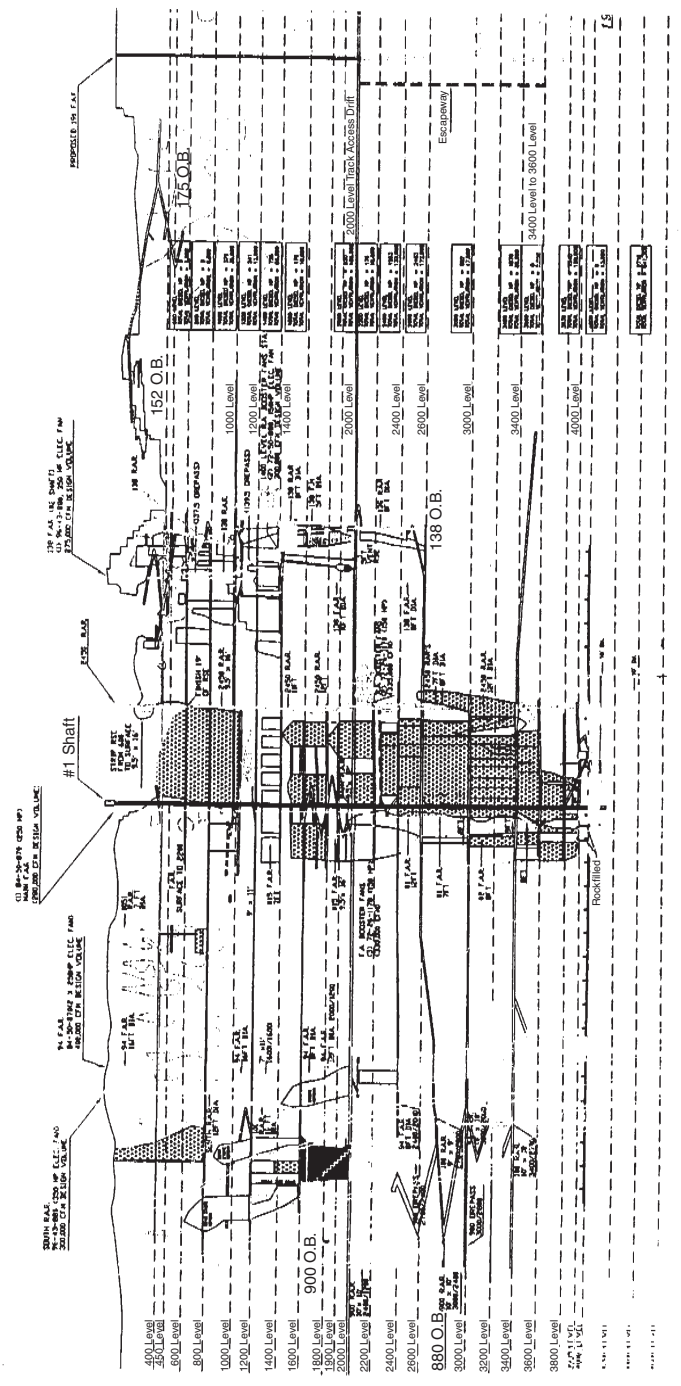


FIGURE 10.1 Longitudinal section—Copper Cliff North Mine

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

200 mm diameter holes. Although mining operations went well, stope filling with rock fill did not. The fill system was not able to keep up with the mining; hence, a number of stopes were always open and waiting for fill.

In January of 1986 during the mining of the 113 stope at the extreme south end of the orebody, there was a dramatic increase in the level of seismicity and rockbursting. This stope was the last primary stope to be mined on this horizon; hence, its strike length was extended beyond its original design length.

ROCKBURST INCIDENTS

By the end of January 1986, bursting had spread from the 113 stope to the 114 and 115.5 stope pillars. There were six bursts greater than 2.0 M_n, the largest being 2.8, 2.8, and 2.9 M_n. There was damage found on the top sill access on the 3835 sublevel, the 3600 and 3935 levels, and the 114 and 115.5 stope pillars. It was inferred that the large bursts were pillar failures. Figure 10.2 is a longitudinal section of mining in the 120 orebody between the 3000 and 3935 levels.

Because of continued high seismicity associated with mining in the 120 orebody a microseismic system was installed to monitor this activity. Mining continued through 1986 with continued microseismic activity but no further bursting. Dilution in the open stopes became a factor as fill delays continued. The failure of the 114 and 115.5 pillars increased loads on the 116.5 and 118.5 pillars, which had already been drilled off. These pillars were blasted into the open 117 stope in December 1986 and March 1987. As a result of this mining, a 2.2 M_n burst occurred on February 13 and a 2.9 M_n burst on March 28, 1987. The mining and bursting resulted in a large cave forming above the mined-out stopes. Mining and bursting continued through 1987 and 1988. There were six more bursts greater than 2.0 M_n in 1987, and six bursts in 1988 were greater than 2.0 M_n, including a 3.3 M_n burst on December 13. Bursting began to occur up on the 3400 level, likely as a result of continued caving. The 3.3 M_n burst displaced about 1,000 t of rock in the 115 section area on the 3400 level. There was less bursting during 1989, as only four bursts greater than 2.0 M_n were reported, but one of these was another 3.3 M_n burst. This burst was not located and no damage was reported or found.

As a result of the bursting in pillars between stopes and the caving problem, the mining geometry was changed to a center-out sequence of stope panels above the 3000 level. There was still bursting in stope pillars below the 3000 level. Though there were no large bursts in 1990, there were five large bursts in 1991, with three of these bursts greater than 3.0 M_n. These large bursts occurred between the 3000 and 3400 levels, primarily in pillars, and resulted in about 2,500 t of damage.

From 1992 to through 1997, mining induced seismicity, but no bursts greater than 1.9 M_n. However, on March 8, 1998, a 3.9 M_n burst occurred in the 120 orebody resulting in more than 3,000 t of damage on a number of levels. And, on May 5, 1998, a 2.8 M_n burst did more than 1,300 t of multilevel damage, also in the 120 orebody. It is not clear if these bursts were associated with current mining or occurred in waste pillars between the 3945 and 2200 levels. Through the year 2000 there were only two bursts greater than 2.0 M_n.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

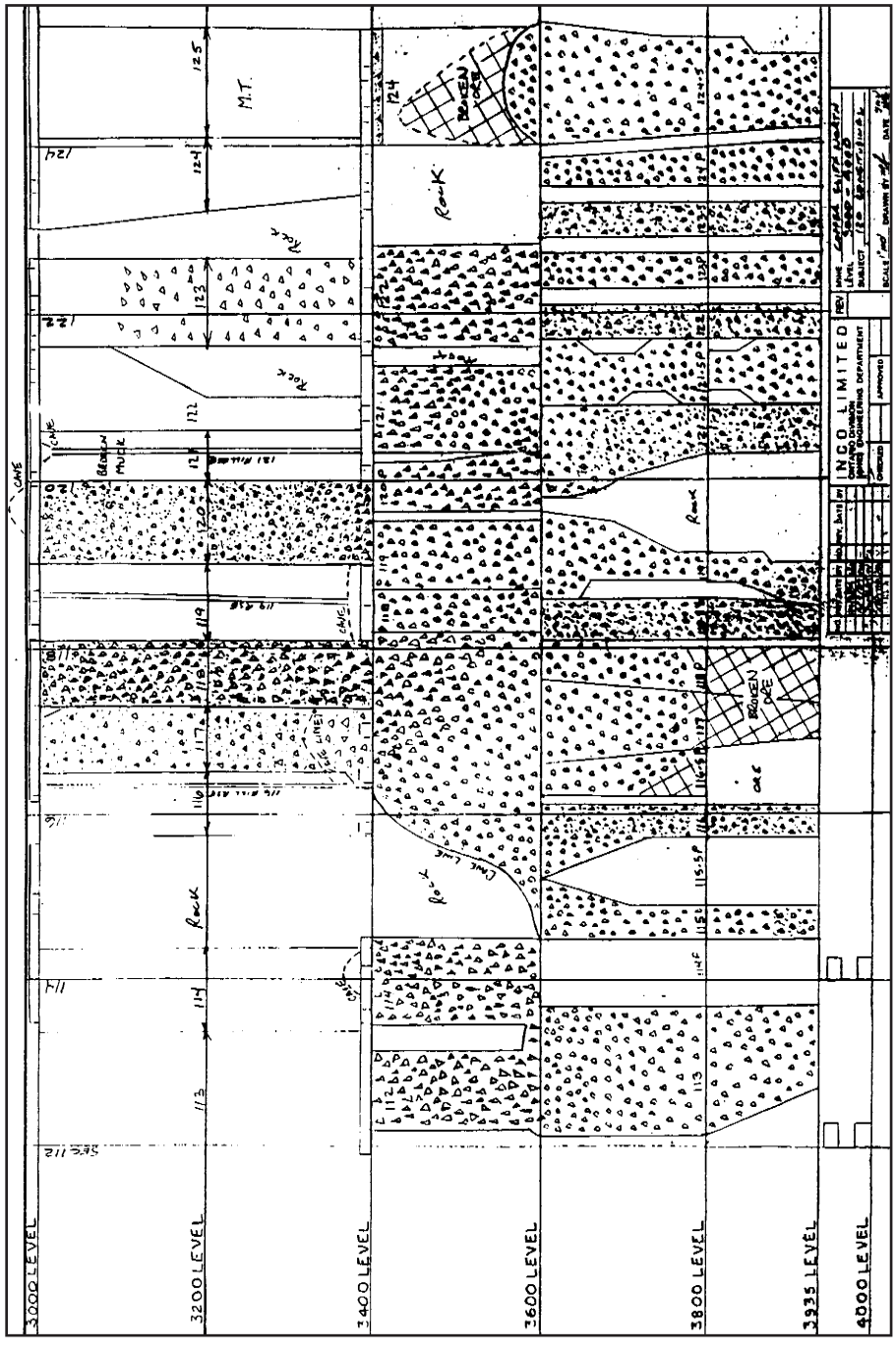


FIGURE 10.2 Longitudinal section 120 orebody

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

DISCUSSION

The majority of rockbursts at Copper Cliff North were the result of the checkerboard pattern of panels initially mined in the 120 orebody. When this geometry was changed to center-out mining panels, the mining-induced bursting decreased dramatically. The 3.9 M_n burst appears to have been the result of the overall mined-out area and the waste pillars left.

As a result of the number and magnitude of bursts greater than 3.0, Copper Cliff North became the second most seismically active mine in the Sudbury Basin after Creighton. The seismicity at Copper Cliff North is not concentrated on well-defined geologic structures as is the case at most other mines in this basin. The seismicity appears to be associated with pillars created or left as a result of mining. By eliminating pillars in mine design the rockburst problem was controlled.

REFERENCES

Morrison, D.M., and J.E. Galbraith. 1990. A case history of Inco's Copper Cliff North Mine. In *Rock Mechanics Contributions and Challenges*. Edited by W. Hustrulid and Johnson. Rotterdam: A.A. Balkema. 51–58.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 11

Quirke Mine
Elliot Lake, Ontario

SUMMARY

Rockbursts at Quirke Mine were a classic example of a chain reaction of pillar failures. Over a 5-year period, 136 rockbursts occurred, of 2.0 up to 3.4 M_n, plus thousands of smaller seismic events. An area larger than 70 hectares (ha) was affected underground. Seismic activity decreased substantially when the hanging wall above the affected area fractured through to surface. This relieved the pressure on the abutments and the area stabilized.

GEOLOGY AND MINING METHODS

The uranium-bearing, quartz-conglomerate reefs at Elliot Lake are in the form of a syncline. At Quirke Mine, the 2–5 m thick reefs dip at about 20° to the south and extend from just below the surface to a depth of 1,000 m. Massive quartzite forms the footwall and hanging wall of the reefs.

Two variations of a stope-and-pillar mining method were employed. In the main reef, rib pillars were systematically laid out on dip at 25 m centers, with crown and sill pillars on strike at the top and bottom of the stopes, typically 100 m long. The blasted ore was scraped to the bottom of the stopes, where boxholes connect to a haulage level 6 m beneath the sill drift.

In the upper reef, about 35 m above the main reef, a trackless mining method was employed. To accommodate the trackless equipment, rib pillars up to 150 m long were laid out at an apparent angle to dip, to ensure a gradient of less than 10°. The two reefs only partially overlap.

ROCKBURST SEQUENCE

A plan of the eastern part of the main reef at Quirke Mine and the adjacent Denison Mine is shown in Figure 11.1. At Quirke, this area of the mine was extracted in the 1970s and early 1980s with about 80% extraction. Between the 7 and 8 levels, a local roll in the reef produced a flat-lying area, which was difficult to recover with slushers. This area was converted into a trial trackless area, prior to the introduction of this layout in the upper reef. The 3-m-wide rib pillars were laid out at an angle of about 45° to accommodate the

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

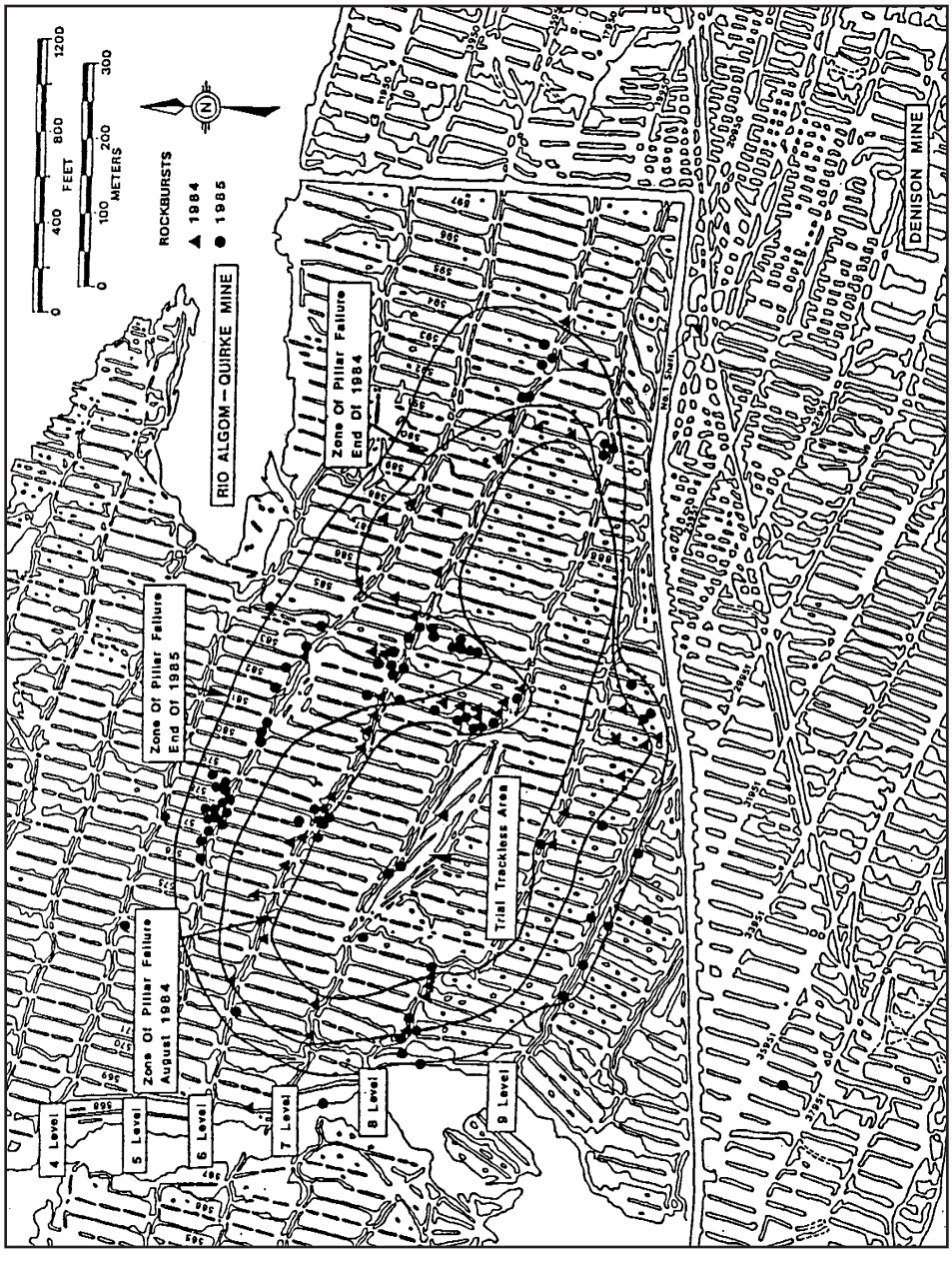


FIGURE 11.1 Plan of Quirke Mine showing location of rockbursts in 1984 and 1985

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

equipment. Mining in this area was started in 1977 and was completed in 1978, with no significant ground control problems.

By 1981 the stopes on the 9 level had been completed. At this time, deterioration, in the form of pillar spalling and floor cracks, was observed in the trial trackless area and the 7 level sill drift immediately up-dip. During the next year the deterioration intensified, accompanied by audible microseismic activity and broken rock bolts. By March 1982 an area 250 m in diameter centered on the trial trackless area was affected.

The first rockburst occurred in the evening of March 12, 1982, and was followed within an hour by four more large bursts, the largest of 3.0 M_n. Initial inspection of the area revealed that both the 7 sill and haulage drifts were blocked off over a distance of 360 m, adjacent to the trial trackless area. A later inspection indicated that the relatively squat sill and crown pillars had failed violently, whereas the slender rib pillars in the stopes, and especially in the trial trackless area, had yielded and failed nonviolently. Floor cracks up to 20 cm wide were observed in the trial trackless area. No significant damage was observed in the roof. However, hundreds of broken rock bolts were observed lying on the floor.

A 32-channel microseismic system was installed around the affected area, with eight sensors on the Denison side of the boundary pillar, to provide better coverage in this area. Between March 1982 and September 1984, the affected area gradually expanded with microseismic activity but no major rockbursts. During this period, access was lost to most of the 7 level and part of the 8 level. By August 1984, the zone of pillar failures extended about 870 m on strike and by 300 m on dip, as shown in Figure 11.1.

Major rockburst activity again occurred in September 1984, and over a 12-month period 154 large seismic events were recorded. The locations of these events are also plotted in Figure 11.1. In September 1984, most of the bursts were concentrated on the 6 sill level directly above the trackless area. In December 1984, seismic activity shifted to some large pillars just northeast of the trackless area, then to the eastern boundary just north of Denison's No. 1 Shaft. By February 1985, the 5 sill level was destroyed, followed by the 9 sill level in August and September of 1985. At the end of 1985, the final zone of pillar failures extended about 1,100 m on strike by some 600 m on dip, as shown in Figure 11.1.

The main pattern of rockburst activity was progressive, violent failure of the pillars at the edge of the affected area. However, some large events were located near the trial trackless area where pillars had already failed. It was suspected that these seismic events were occurring in the hanging wall. This was confirmed in April 1985 when the water flow into the rockburst area suddenly increased to 1,000 L/min, and a small lake directly above the area drained into the mine. Subsequently, diamond drill holes from surface into the rockburst area encountered lateral slippage at bedding contacts, gaps up to 150 mm wide, and loss of circulation water at numerous horizons. This indicated that the immediate hanging wall above the affected area had probably caved to within 160 m from the surface. The ground above this had fractured through to surface, although there were no visible signs of surface subsidence.

With the breakdown of the hanging wall seismic and microseismic activity substantially decreased. The affected area stopped expanding and stabilized, with little further damage being reported. In the following years, until the mine closed in 1990, the occasional rockburst was recorded and thought to be further adjustments in the hanging wall.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

DISCUSSION

The cause of the rockbursts at Quirke Mine was the design of the rib pillars in the trial trackless area. Being laid out at an angle of about 45° to true dip, these pillars had less cross-sectional area for the stress to pass through, between the hanging wall and footwall. Hence, they were weaker than the normal rib pillars on dip, with a safety factor of about 1.0.

Initially the underdesign of these pillars was not obvious, because the area was near the solid abutments. However, 3 years later the trackless area was near the center of the mine workings, and the rib pillars were subjected to the full load associated with 80% extraction. The strength of these pillars was exceeded, they failed nonviolently, yielded, and transferred load to the stronger sill and crown pillars on the 7 level. Eventually, the stress on these pillars exceeded their strength and they failed violently. These bursts initiated the chain reaction of further load transfer and an expanding area of pillar failures. This process continued until the hanging wall could no longer span the affected area. The critical span was exceeded in April 1985, when there was a large water inflow into the mine. At this time the minimum span of the affected area was about 400 m on dip. The average depth of the area was 500 m, giving an aspect ratio of 0.8. This ratio is high compared to other mines subject to caving. It is probably due to the ability of the massive quartzite beds to span large distances without major breakdown.

The chain reaction of pillar failures at Quirke was relatively slow, lasting almost 5 years. This is attributed to the variation in pillar size and strength. It was observed that the affected area rapidly expanded through the stoping areas with slender rib pillars. Then seismic activity and rockbursts would concentrate on the sill pillars for some months before the area would expand again. If the pillars had been more of a uniform size, the chain reaction would have been much more rapid.

When rockbursts occur in an open stope-and-pillar layout, there is very little that can be done other than abandon the affected area while seismic activity continues. The one stabilizing measure is to pour cemented backfill into the affected area, an option not available to Quirke Mine. However, the adjacent Denison Mine did have a backfill plant and started to backfill the area of their mine adjacent to the boundary pillar and directly down-dip from the rockburst area at Quirke Mine. Two facts were noticeable. When hydraulic backfill was poured into an area of highly stressed pillars, microseismic activity increased and there was the occasional rockburst in the area. However, once the pillars were totally encased in backfill, seismic activity ceased.

The more conventional approach for preventing rockbursts in stope-and-pillar layouts is carried out during the design and planning stages. Pillars should be designed with a minimum safety factor of 1.5 to prevent any failure. Alternatively, barrier pillars at regular intervals should contain and isolate areas of pillar failure. This latter approach was used at depths below 1,000 m at the Stanleigh Mine in Elliot Lake.

The rockbursts at Quirke readily indicated which ground support systems were ineffective and which survived dynamic loading. Forged-head mechanical bolts in the drifts, pillar sides, and stope backs readily failed. Similarly, splitsets by themselves were ineffective, with the rock unraveling behind the plates. Soon after the start of rockbursts in March 1982, an attempt was made to rehabilitate the 8 haulage level with alternating grouted rebar and splitsets with chain link mesh. This haulage drift was subjected to rockbursts and the rehabilitated section survived, with some bagging of loose rock contained by the mesh and noticeable slippage on the splitsets. The sections of the drift still supported by conventional mechanical bolts totally collapsed.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

With a chain reaction of pillar failures, there is normally advanced warning of impending rockbursts. At Quirke Mine the first signs were audible working of the ground and broken bolts, prior to any deterioration of pillars. Then pillar spalling commenced, especially at the corners of the roof and pillar, followed by sloughing off of the pillar walls. Microseismic activity also tended to increase before rockburst activity. Prior to the rockbursts in September 1984, microseismic activity progressively increased over an 8-month period when plotted on a monthly basis. With shorter time periods (i.e., hours, days, or weeks), the plots were much more erratic and inconclusive.

REFERENCES

Hedley, D.G.F. 1992. Rockburst handbook for Ontario hardrock mines. *CANMET Special Report SP92-1E*. Ottawa: CANMET, Energy, Mines and Resources, Canada.

Hedley, D.G.F., J.W. Roxburgh, and S.N. Muppalaneni. 1984. A case history of rockbursts at Elliot Lake. *2nd International Conference Stability in Underground Mining*, Lexington, Kentucky.

Hedley, D.G.F., and N. Whitten. 1983. Performance of bolting systems subject to rockbursts. *CIM Seminar Underground Support Systems*, Sudbury. CIM Special volume 35. 73–79.

MacDonald, P., and S.N. Muppalaneni. 1983. Microseismic monitoring in a uranium mine. In *Rockbursts: Prediction and Control*. London: Institution of Mining and Metallurgy. 141–145.

Swan, G. 1985. Strength distributions and potential for multiple pillar collapse. *Division Report MRP/MRL 85–87*. Ottawa: CANMET, Energy, Mines and Resources, Canada.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 12

Campbell Mine Red Lake, Ontario

SUMMARY

Gold mines in the Red Lake area have a history of rockbursts dating back to the mid-1960s. At the Campbell Mine the two ore zones most affected by bursts were the A and F zones. Bursting in the A zone began in 1965 during sill and boxhole pillar mining. At the end of 1983, a significant rockburst series occurred in the F zone. This ore zone was being mined by shrinkage stoping to a depth of about 650 m, leaving small boxhole and thin horizontal sill pillars. Over a 30-hour period, 22 large rockbursts, up to 3.3 M_n, were recorded, as well as hundreds of microseismic events. These bursts were caused by a relatively fast chain reaction of sill and boxhole pillar failures over 7 levels. The area affected underground was about 450 m on strike by 300 m on dip. Since that time, bursting at Campbell has mostly been associated with sill pillar mining.

GEOLOGY AND MINING METHODS

Campbell Mine is situated at Balmertown in the Red Lake District of Northwestern Ontario. The ore at Campbell occurs in steeply dipping gold-bearing veins in a folded sequence of Precambrian volcanic rocks. There are two main vein structures: quartz carbonate fracture-filled veins 0.2–1.0 m wide within the andesite rock and replacement veins 0.6–9.0 m wide along the contact between the andesite and altered rock units. The faults along which the vein formed were rehealed during mineralization. Some minor postmineralization faults offset the veins, and recently, a number of shear zones have been identified which are primarily oriented parallel to the ore zones.

The mine was brought into production in 1949. The first shaft was deepened in stages to 1,315 m with levels established at 45 m intervals. The new Reid Shaft was sunk to 1,873 m and currently services the mine on 27 and 30 levels. Initially, all production was from shrinkage stopes, but in the 1960s some of the ore zones were converted to overhand cut-and-fill. In the late 1980s sublevel long-hole mining was first used to recover sill pillars, then became a primary mining method. Daily production, some 1,200 t, presently consists of 64% long-hole mining and 36% cut-and-fill.

After bursting began to occur in the A zone in 1965, as well as in the adjacent Dickenson Mine, both companies retained Professor R.G.K. Morrison of McGill University, Montreal, to study the rockburst problem. His recommendations included increasing the size of

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

sill pillars and mining shorter stopes, rather than continuing with the existing long flat backs in shrinkage stopes. He also suggested elimination of boxhole pillars in favor of timbered backs in new shrinkage stopes and conversion to cut-and-fill mining methods at depth. In the A ore zone at Campbell, these recommendations were followed and conversion to cut-and-fill mining was started below 10 level. However, the F zone continued to be mined with shrinkage stopes, boxhole pillars, and a standard size for sill pillars.

ROCKBURST SEQUENCE IN THE F ZONE

The F zone is an isolated orebody at the west end of the mine. Just below the surface the strike length is about 450 m, which tapers down to 150 m on 15 level. Vein thickness varies from 1.2–3.0 m, with several offshoots also mined. The dip is generally 75° to the south.

Mining of the F zone began on the upper four levels in the 1950s and proceeded downwards as the shaft was deepened. By 1974, essentially all the ore above 10 level had been mined by shrinkage methods with boxhole pillars and 6 m sill pillars below each level.

Deterioration and minor bursting started to occur along old, mined-out levels in the early 1980s. The first burst in 1981 occurred at the west end of 11 level. Over a 30-month period, additional bursts were located in boxhole pillars between 7 and 14 levels. These bursts were triggered either by drawdown of broken ore in the stopes or renovation work on the levels and, in one case, destress blasting of a few boxhole pillars on 14 level.

In August 1983, a 32-channel microseismic system was installed underground, with 13 channels surrounding the F zone. The last warning in this zone was on December 20, 1983, when a renovation crew was removed from 12 level because of seismic activity and deterioration of boxhole pillars.

At 12:51 a.m. on December 30, 1983, the first violent rockburst was felt on surface, followed rapidly by other large events. The microseismic system indicated that bursting was confined to the F zone, where no crews were working. However, work crews in other parts of the mine felt the vibrations and were evacuated to surface.

Figure 12.1 shows the stope and pillar layout in the affected area and the sequence and location of the major bursts, including 22 bursts with an assigned magnitude. Rockbursts were initiated on 10 level and within minutes had spread to 11 level, and within an hour to 9 level. By 8:40 a.m. on December 30, rockbursts had spread to 8 level. For the rest of the day, bursting was mainly confined to 8, 9, and 10 levels, and after a flurry of rockbursts at 2:30 a.m. on December 31, activity ceased on these levels. For the rest of December 31 and January 1, activity was mainly on 7, 12, and 13 levels. Sporadic activity continued after this, but mainly along the fringes of the affected area. Figure 12.1 also indicates that the vast majority of the major rockbursts occurred where sill pillars were 6 m or less. Very few events occurred where sills were greater than 10 m. Additionally, the first rockburst on each level tended to occur where a service raise passed through the sill pillar, which created a double-stress concentration.

The first visual inspection of the F zone was done on January 5, 1984. It was found that 7 to 13 levels were blocked off near the eastern boundary of the ore zone. Minor damage had occurred on 14 level, and no damage on 15 level. Levels 4, 5, and 6 were open with only some loose shake-down and fresh cracks.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

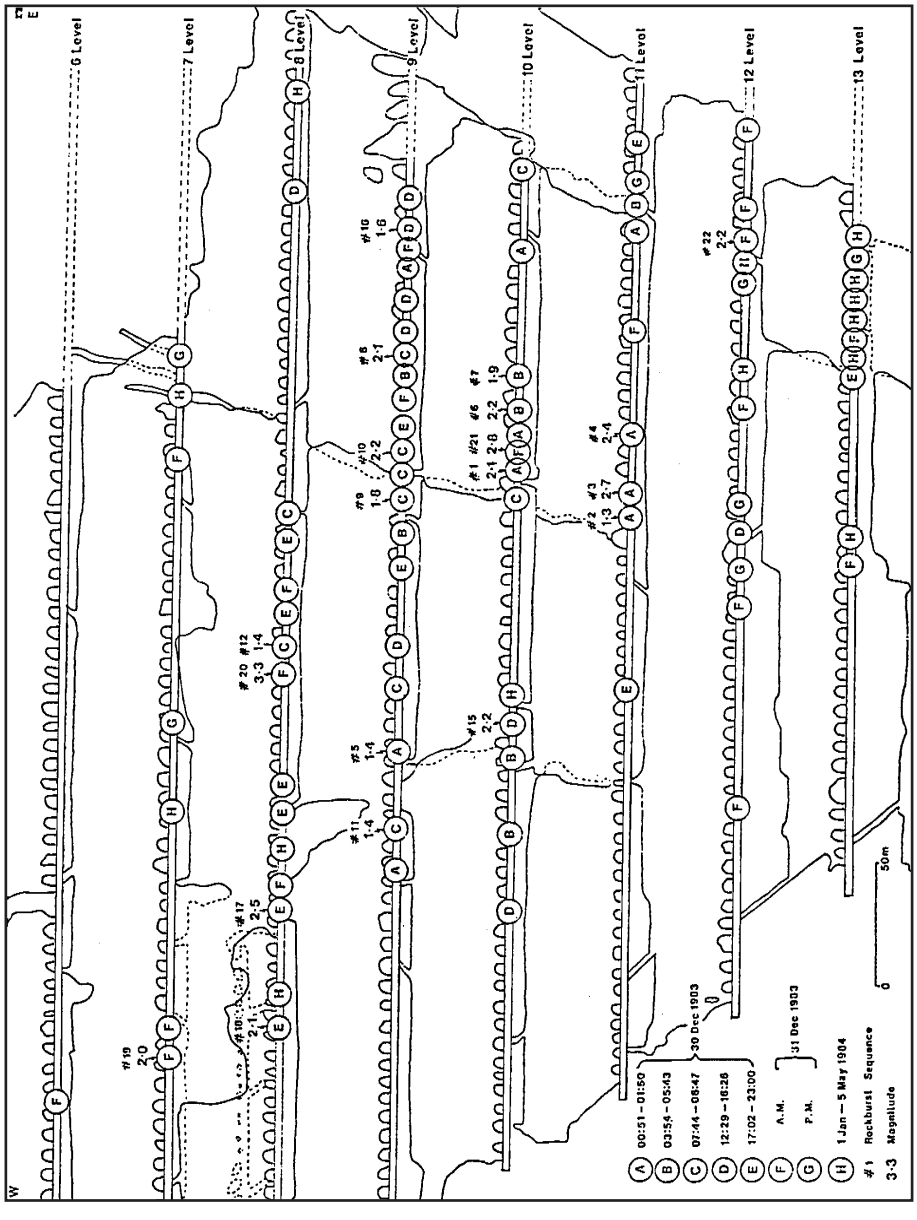


FIGURE 12.1 Longitudinal section of F zone showing location and sequence of major seismic events

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

In October 1984, an inspection was made of the west end of the F zone on the 12, 13, and 14 levels. The west end of 14 level was relatively undamaged over a distance of 100 m. Similarly, the last 75 m of 13 level was relatively undamaged, whereas heavy spalling had occurred along the complete length of 12 level. The backs along the shrinkage stopes on 13 and 14 levels were open and undamaged where the sill pillars were 15–18 m but showed signs of high stress where the sill pillar narrowed down to 6 m.

In the early 1990s a hanging wall bypass drift was driven around F zone for exploration purposes. Inspection of the boxhole pillars on 14 level indicated no further deterioration; hence, it was decided to recover the broken ore along the west end of the 1421 stope, where the sill pillar was 15 m. From a footwall bypass drift starting at the west end of the stope, crosscuts were driven to intersect the stope and allow for extraction of the broken ore. This development work was successfully carried out and the broken ore extracted with little microseismic activity and no rockbursting.

ROCKBURSTS IN THE A AND OTHER ZONES

Rockbursts in the A and other zones at Campbell have essentially been associated with sill pillar recovery mining. A minimum 10 m sill pillar is now left above cut-and-fill stopes and later recovered by long-hole mining, primarily using upholes drilled from along the last stope lift. Most rockbursts induced have been the result of pillar-type geometries created by stress interaction with old stope raises or an incomplete drilling pattern where drill access was restricted. Rockbursting associated with primary sublevel long-hole mining has occurred along stope abutments as well as where small pillars may have been left or created.

There have been minor burst sequences in old, mined-out shrinkage areas of both the A and F ore zones, triggered by mining adjacent and parallel footwall veins. In March of 1986 cut-and-fill mining of a footwall vein on 8 level resulted in a sill pillar burst along the mined-out 9 level. This was followed by seven more bursts in remnant boxhole or sill pillars between 8 and 10 levels during the next hour. The three largest of these bursts were in the 2.5 M_n range. After a week's quiet period, two more bursts occurred on 8 and 9 levels, ending this burst sequence. There was no damage or indication of stress in the stope being mined, which was in the stress shadow of the mined-out A zone. An almost identical burst sequence occurred in boxhole and sill pillars along 12 and 13 levels in the F zone from mining parallel footwall veins on 10 and 11 levels.

DISCUSSION

When broken ore is completely removed from shrinkage stopes, the stope geometry becomes an open stope and pillar layout similar to the Elliot Lake uranium mines. Under these conditions the pillars have to be able to support the load imposed by the hanging wall and footwall without failing. In the case of the F zone, this would have meant leaving progressively larger sill pillars as the mining became deeper. Alternatively, the orebody could have been converted to cut-and-fill mining, similar to the A zone. Rockbursts still may occur in the sill pillar of cut-and-fill stopes, but they tend to be isolated occurrences.

After the failures numerical modeling of the F zone indicated that the boxhole pillars from 9 level down had probably yielded and shed load to the sill pillars. Both the sill pillars on the 10 and 11 levels had a safety factor of about 1.0, and this, in fact, is where the rockburst sequence started.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

The chain reaction of pillar failures in the F zone occurred over a 30-hour period, compared to almost 5 years at Quirke Mine. The area affected was smaller, being 13 ha at Campbell and 70 ha at Quirke, and the pillars that failed were of a more uniform size. It is unlikely that the hanging wall of the affected area caved and fractured similarly to Quirke, since later attempts to mine a footwall vein on both 10 and 11 levels induced bursting on 12 and 13 levels in the F zone.

Although the microseismic system indicated no abnormal activity prior to the rockburst sequence, there was a 30-month history of deteriorating conditions including minor bursting. Further deterioration was anticipated, and the crews had been removed from all mining and renovation work being carried out in F zone some weeks prior to the major rockburst sequence.

The rockburst strategy of leaving larger sill pillars in cut-and-fill stopes, then recovering these pillars by long-hole mining, without reducing the pillar size, has been found to work well at Campbell. Converting from cut-and-fill to primary long-hole mining, where possible, has also improved rockburst control. However, primary long-hole mining has resulted in inducing bursts in adjacent and nearby mining zones. And, the larger blasts and faster mining rate has also resulted in inducing bursts on structures out in the walls. This has been a departure from the traditional bursts in small pillars in the plane of the ore.

REFERENCES

Hedley, D.G.F. 1992. Rockburst handbook for Ontario hardrock mines. *CANMET Special Report SP92-1E*. Ottawa: CANMET, Energy, Mines and Resources, Canada.

Hedley, D.G.F., M. Neumann, T. Makuch, and W. Blake. 1985. Rockbursts at Campbell Red Lake Mine. *CIM Rockburst Seminar*, Sudbury, Ontario.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 13

Lucky Friday Mine Mullan, Idaho

SUMMARY

As bursting increased in the late 1960s, the mining geometry at Lucky Friday was changed from a single, flat-backed overhand cut-and-fill mining method to a center lead stope mining geometry. When the mining front entered the Lower Revett Formation in the early 1980s, sill pillar bursting became such a serious problem that the mine had to convert to an underhand cut-and-fill longwall geometry to improve safety. While bursting has continued, the mechanisms of bursts have changed, including an increase in the number of very large bursts. Lucky Friday has been able to manage their rockburst problem and mine more safely and efficiently.

GEOLOGY AND MINING METHODS

The steeply dipping silver-lead veins at the Lucky Friday occur in and intersect the quartzites and argillites of the Precambrian Belt Series. This bedded sequence ranges from hard, vitreous, and sericitic quartzites up to 1.5 m in thickness to soft and deformable argillites, often less than 100 mm thick. The original bedded sedimentary rocks have undergone some five periods of tectonic deformation resulting in a rock mass that is highly folded and faulted. Ore deposition likely took place during the third period of tectonic activity.

The strike length of the Lucky Friday Main Vein is about 500 m, and vein thickness varies from zero to about 6 m. The vein dips more steeply than the bedding, both dipping to the south. Three major faults intersect the vein with up to 4 m of lateral displacement. The North and South Control faults appear to limit the vein, while the Offset Fault occurs near the center of the vein. The strike length of the vein on surface was very short and did not widen out until the hard quartzites of the Revett Formation were encountered at a depth of about 400 m. Mining is currently being carried out on the 5930 sublevel, some 1,808 m below surface.

Initial mining began in 1941 on the 300 level using overhand cut-and-fill. Below the 1800 level the vein rapidly increased to its full strike length, and the No. 2 Shaft was completed to 964 m in 1957 to allow for increased production. Flat-backed overhand cut-and-fill was the mining method. By 1982 production had increased to about 950 t per day, and in 1983 the Silver Shaft was completed to a depth of 1,890 m to allow for

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

lower-level mining. In 1986 the mine began conversion to underhand cut-and-fill mining to deal with a serious and increasing rockburst problem.

ROCKBURST INCIDENTS

The rockburst incidence increased in the 1960s as each flat-backed stope was mined up to and through a sill pillar to achieve 100% extraction. Figure 13.1 shows a longitudinal projection of the mining on the Lucky Friday Vein. Large bursts, generally in the 2.0- to 2.5-M_l range, caused widespread damage to a number of stopes. Therefore, the mining geometry was changed to a center, lead stope, pyramid shape that limited sill pillar bursts to a single stope. In 1973 a 24-channel microseismic monitoring system was installed and used to determine when a stope pillar became stressed enough to require pillar destressing. By 1975 it became apparent that burst incidences were decreasing, and seismic velocity surveys through sill pillars indicated that pillars were self-destressing, eliminating the need for pillar destressing. It was later determined that this was caused by the mining front entering the Middle Revett Formation, which was less brittle and tended to yield rather than burst.

In 1982 the incidence of rockbursts increased dramatically and continued to increase. Large bursts were disrupting production by Wednesday of a workweek, and bursts were also occurring during first-floor mining in stopes on the 4900 and 5100 levels. A new 32-channel interactive microseismic system was installed, and stope pillar destressing was resumed. In 1985 a mechanized undercut-and-fill test stope was started at the east end of the mine on the 5100 level. A paste-type backfill was developed and utilized to provide a stable stope back. Stratigraphic mapping revealed that the increased bursting was due to the mining front entering the Lower Revett Formation. The quartzites of this formation are much harder and contain less argillitic interbeds than those of the overlying Middle Revett. By 1986 the burst problem had become so severe that miners were becoming hesitant to go underground.

In April of 1986 the Lucky Friday was shut down due to low metal prices. Mining was carried out, however, in the underhand stope to allow continued testing of this mining method. Based on favorable test results, the decision was made to convert the entire mine to sublevel undercut-and-fill mining to be able to deal with the rockburst problem.

Upon resumption of mining in October 1987, a second underhand stope was phased in just west of the initial underhand stope. To maintain production, remnant pillar mining was carried out above the 4900 level by undercut-and-fill mining. This required the remnant sill pillars to be destressed by drilling and blasting a single row of large-diameter downholes along the vein. By the early 1990s, conversion to underhand mining was completed. It should be mentioned that remnant pillar mining was abandoned when some large bursts, up to 3.6 M_l, caused excessive damage to stope and development access openings.

Full production from five mechanized underhand stopes was some 550 t per day, in contrast to the previous 950 t per day from more than 20 overhand slusher stopes. Three stopes were on the Lucky Friday Main Vein, one on a footwall vein split, and one on the Silver Vein, which was located at the west end of the Main Vein.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

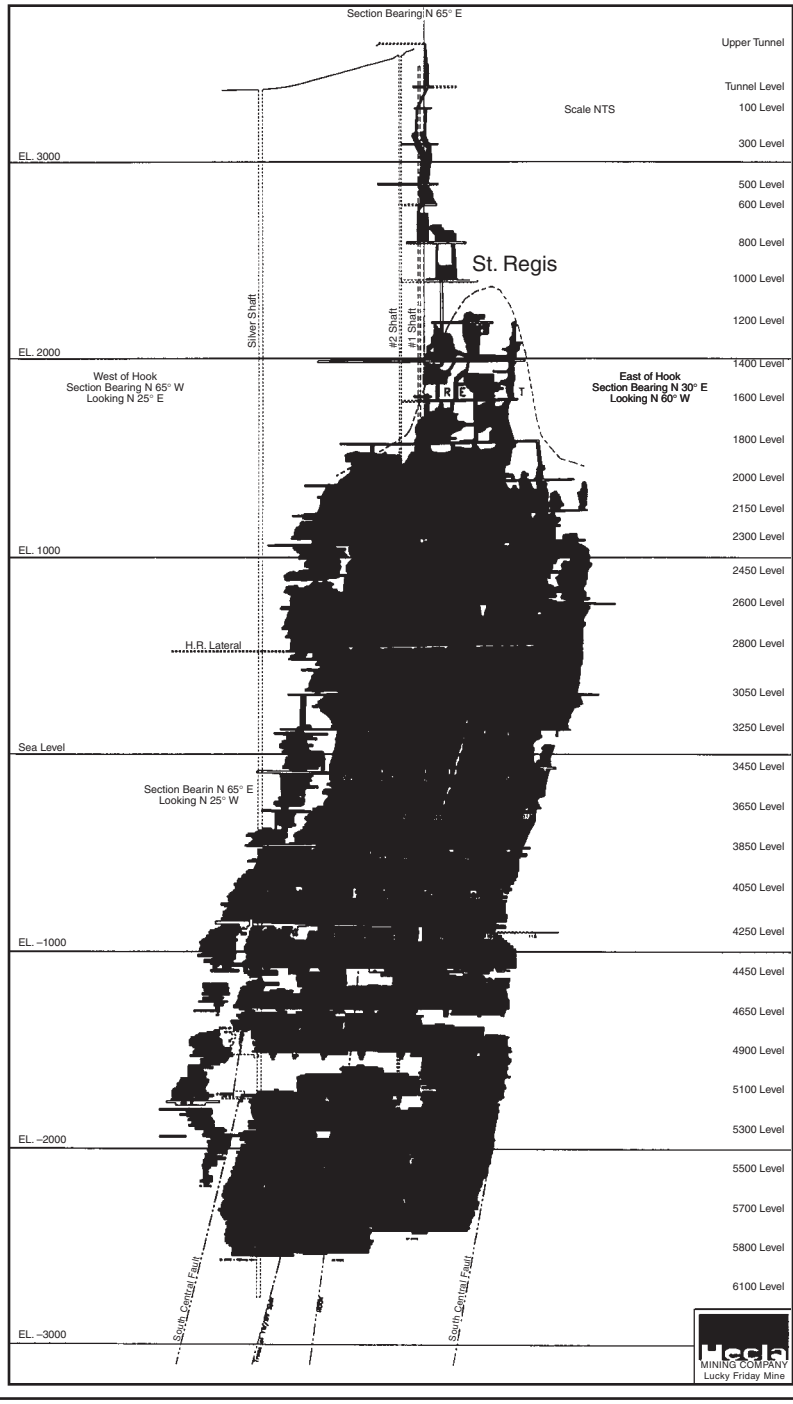


FIGURE 13.1 Generalized longitudinal projection—Lucky Friday Mine

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

Productivity doubled as a result of underhand mining, costs dropped significantly, and safety improved dramatically. However, despite not creating sill pillars, rockbursting continued, and rockburst magnitudes have been larger with underhand mining.

Rockbursts with Underhand Mining

The change to underhand mining at the Lucky Friday also changed the burst mechanism. The predominant type of bursting induced by overhand mining was pillar bursting, however, the bursts primarily occurred as a result of slip on a favorably oriented bedding plane that intersected the pillar. Most pillar bursts were in the 2.0- to 2.5-M_l range. Prior to underhand mining only two bursts were larger than 3.0 M_l. A 3.5 M_l burst occurred in 1969 when a flat-backed mining front was still being used, and a 3.1 M_l pillar burst occurred in 1984 in a center lead stope. Since conversion to underhand mining in 1987, there have been 25 bursts greater than 3.0 M_l and two bursts larger than 4.0 M_l.

The first 4.0 M_l burst occurred in October 1988 as the second underhand longwall stope was being started. A vertical stabilizing pillar was being left between stopes, which also happened to contain the Offset Fault. The burst closed off the underhand stope as it approached the stabilizing pillar, just below the 5100 level, and did extensive damage to crosscut access to the old overhand stopes on both 5100 and 4900 levels in the vicinity of the fault. In addition, the raise for the 5100 level overhand stope was blocked off above the level, and the circular, steel-lined mill holes in the backfill showed about 20 cm of stope closure. Observable damage was in the 1,000 t range. Though it appeared, from both its source location and its damage pattern, that this burst occurred along the Offset Fault, a numerical modeling back-analysis of the mining geometry could not produce a burst of this magnitude. There was no anomalous microseismic or seismic activity prior to this burst, and there was an unusually short and quiet aftershock pattern of seismic activity following this burst.

In August of 1994 a 4.1 M_l burst occurred at the Lucky Friday. This was the largest burst in the history of the Coeur d'Alene Mining District. The burst was located on the North Control Fault just below the 5300 level. The closest mining was on the 5570 sublevel where the 07 stope was approaching the intersection of the North Control Fault. Damage in this stope was relatively light, less than 100 t of displaced rock, and was very well contained by the stope wall and back support. The fill was compressed from closure effects all along the stope, with one small area of broken and bagged fill at the slot intersection. All of the damage was contained by the backfill support of Dywidag bolts and chain link mesh. A splay off of the North Control Fault showed some 15 cm of vertical displacement about 20 m from the fault, but this was not considered to be primary fault movement. Above the east end of this stope and just below the 5210 sublevel, 1.7 cm of closure was measured along a drift that had subsequently been mined through the filled vein. On the 5100 level, about 80 m west of the North Control Fault, 6 cm of closure was measured along an unfilled portion of the mined-out vein. Previous major bursts at the Lucky Friday had never resulted in such widespread closure. There was no anomalous seismic activity immediately prior to this burst.

A 10-year review of overall seismicity and mining activity at the Lucky Friday from 1988 through 1998 revealed a 6-month downward trend in the rockburst rate, or released seismic energy, prior to both this and the previous 4.0 M_l event, as is shown in Figure 13.2.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

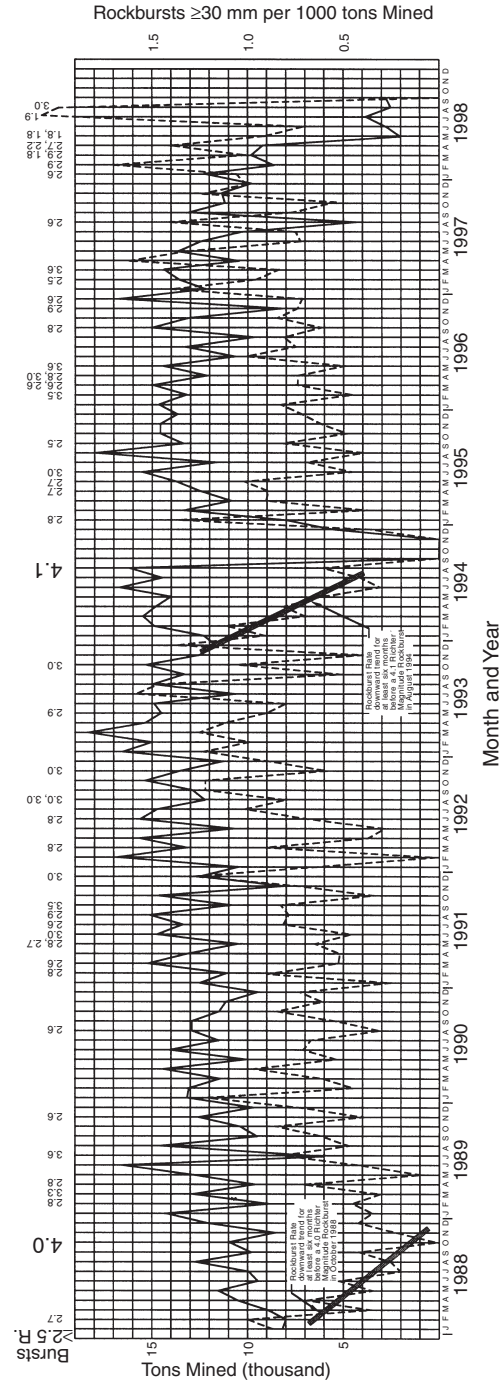


FIGURE 13.2 Rockburst rate versus tonnage

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

DISCUSSION

The mining geometry was changed from a single long flat-back to an advancing pyramidal front to reduce stope strike length and transfer stress more gradually to the abutments. This mining change, along with stope pillar destressing, allowed mining to progress until the Lower Revett Formation was encountered. To insure that mining could continue to be carried out safely, the mining front was converted from overhand cut-and-fill to a mechanized underhand cut-and-fill. This mining method has allowed the mine to deal with a severe rockburst problem while increasing productivity and reducing both costs and accidents. The benefits of underhand mining at the Lucky Friday are evident as shown on Figure 13.3.

At Lucky Friday more than eight bursts per year are in the 2.0- to −3.0-M_l range, and more than one per year larger than 3.0 M_l. Most of these large bursts have occurred out in the hanging wall on the North and South Control faults, with mostly strike slip movement toward the vein. Major bursts also occur as shear failures along bedding planes in the footwall, as well as slippage along the vein itself, primarily vertical offset. The major bursts do not appear to be related to the day-to-day mining, but rather to the overall mine geometry and continuing mine closure. It appears likely that increasing the strike length of a stope from 60 m to 150 m has been responsible for the increased burst magnitudes. The Lucky Friday is the most burst-prone mine in this mining district, and is the only mine that has experienced large rockbursts, greater than 3.5 M_l.

Sill pillar destressing has been very successful at the Lucky Friday. Destress holes were gradually increased in size from 50 mm to 150 mm. The largest pillar destressed was some 180 m long by 30 m high. A single row of 58 holes on a 3 m spacing was drilled to a 24 m depth. The bottom 15 m of each hole was loaded with 118 kg of explosive, 6 m of stemming was added, and shot with millisecond delays. Fluid injection has also been tried to destress a pillar by inducing a burst. However, none of the fluid injection attempts were successful in triggering an immediate seismic response. It was postulated that there was not a single, well-defined structure that was ready to slip.

Stope reinforcement for the old overhand stopes consisted of a combination of timber, rock bolts, and mats. This reinforcement was ineffective in containing the effects of bursting. With underhand mining, stope walls are reinforced with splitsets and chain link mesh, and the fill back is reinforced with vertical Dywidag bolts, chain link mesh, and splitsets as needed. The stope entry slot is similarly reinforced, plus the addition of two or more horizontal rows of cable-type lacing. This reinforcement has been very successful in containing burst damage. Development openings are reinforced with a combination of Dywidag bolts and splitsets and chain link mesh in the back, as well as splitsets and chain link mesh down the walls. The wraps of chain link mesh are prerolled to size and installed at right angles to face advance, starting at the center of the drift or stope back. In severe “bumping” ground a newly blasted development heading is first sprayed with a nominal 50 mm of steel-fiber-reinforced shotcrete and then reinforced with the standard pattern of bolts and chain link mesh. The immediate application of shotcrete has essentially eliminated the small strain bursts that frequently caused injuries during drilling for reinforcement or face advance.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

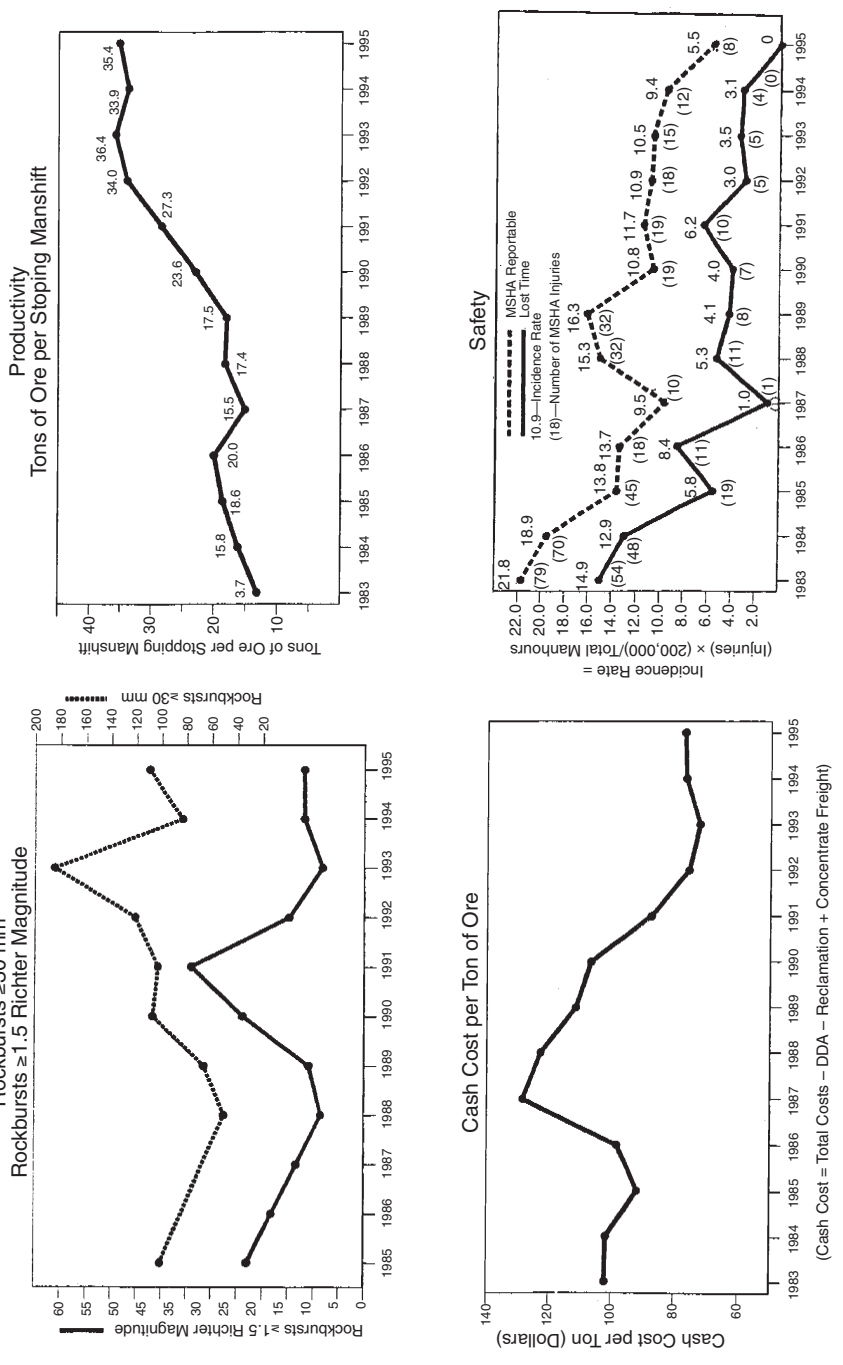


FIGURE 13.3 Improvements from underhand mining at the Lucky Friday Mine

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

REFERENCES

Blake, W., and D.J. Cuvelier. 1990. Developing reinforcement requirements for rockburst conditions at Hecla's Lucky Friday Mine. In *2nd International Symposium Rockburst and Seismicity in Mines*, Minneapolis, MN. 401–406.

Langstaff, J.T. 1976. Hecla's Seismic Detection System. *17th U.S. Rock Mechanics Symp*osium Snowbird, Utah. 3A7-1–3A7-7.

Noyes, R.R., G.H. Johnson, and S.D. Lautenschlaeger. 1988. Underhand stoping at the Lucky Friday Mine. *94th Annual Meeting Northwest Mining Assoc.,* Spokane, WA. Paper No. 2.

Whyatt, J.K., W. Blake, and T.J. Williams. 1997. Classification of large seismic events at the Lucky Friday Mine. *Trans. Institution of Mining and Metallurgy*, Sect. A. A148–A162.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 14

Star Mine Burke, Idaho

SUMMARY

Rockbursts at the Star Mine were primarily associated with initial first-floor mining as well as sill pillar mining. Converting to a center lead stope geometry did not reduce the incidents of bursting; thus, it was decided to destress these two zones during stope development. When this strategy proved to be successful, it was decided to destress an entire stope block prior to mining. This also proved to be successful.

GEOLOGY AND MINING METHODS

The Star Mine was the deepest lead-zinc mine in the world. The ore shoot is along a vertical shear zone, the Morning East Fault, that strikes an average of N75°W and dips 85°N. Maximum strike length is about 1,200 m with an average stoping width of 3 m. The vein is less competent than the wall rocks, which are hard brittle quartzites, argillites, and argillaceous quartzites of the Lower Revett Formation. The vein intersects the near vertical dipping bedding of the wall rocks at angles varying from 30° to 60°. The vein is also cut by a number of basic dykes. The Star-Morning ore shoot, shown in Figure 14.1, was the largest in the Coeur d'Alene Mining District. Ore has been mined on up to 20 separate veins within the ore shoot. Some 24,000,000 t of ore were mined since the Morning Mine started in 1895.

A timbered cut-and-fill mining method was used to support the heavy ground encountered at depth. A blind stoping method was used with cribbed raises carried in the fill. The veins were accessed from lateral drifts driven 20 m to 40 m out in the footwall. Each timbered set consisted of a cap, two posts, and two squeeze headings. The caps were round tree sections from 0.3 m to 0.6 m in diameter depending on stope width. The squeeze headboards were made up of short sections of lagging and could be up to 0.75 m in length. Lagging from the caps to the back was used to support the heavy and stress-fractured ore. Walls were supported with bolts and mats as required. Stoping was carried out in 3 m lifts using horizontal breast rounds. Once a floor was complete, the raise-up for the next floor was mined, the crib was raised, the mined-out floor sanded, then mining began on the next floor.

As can be seen in Figure 14.1, the mine was accessed by a number of tunnels, shafts, and winzes. The No. 4 Shaft was sunk in the late 1960s from the main 2000 level adit to

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

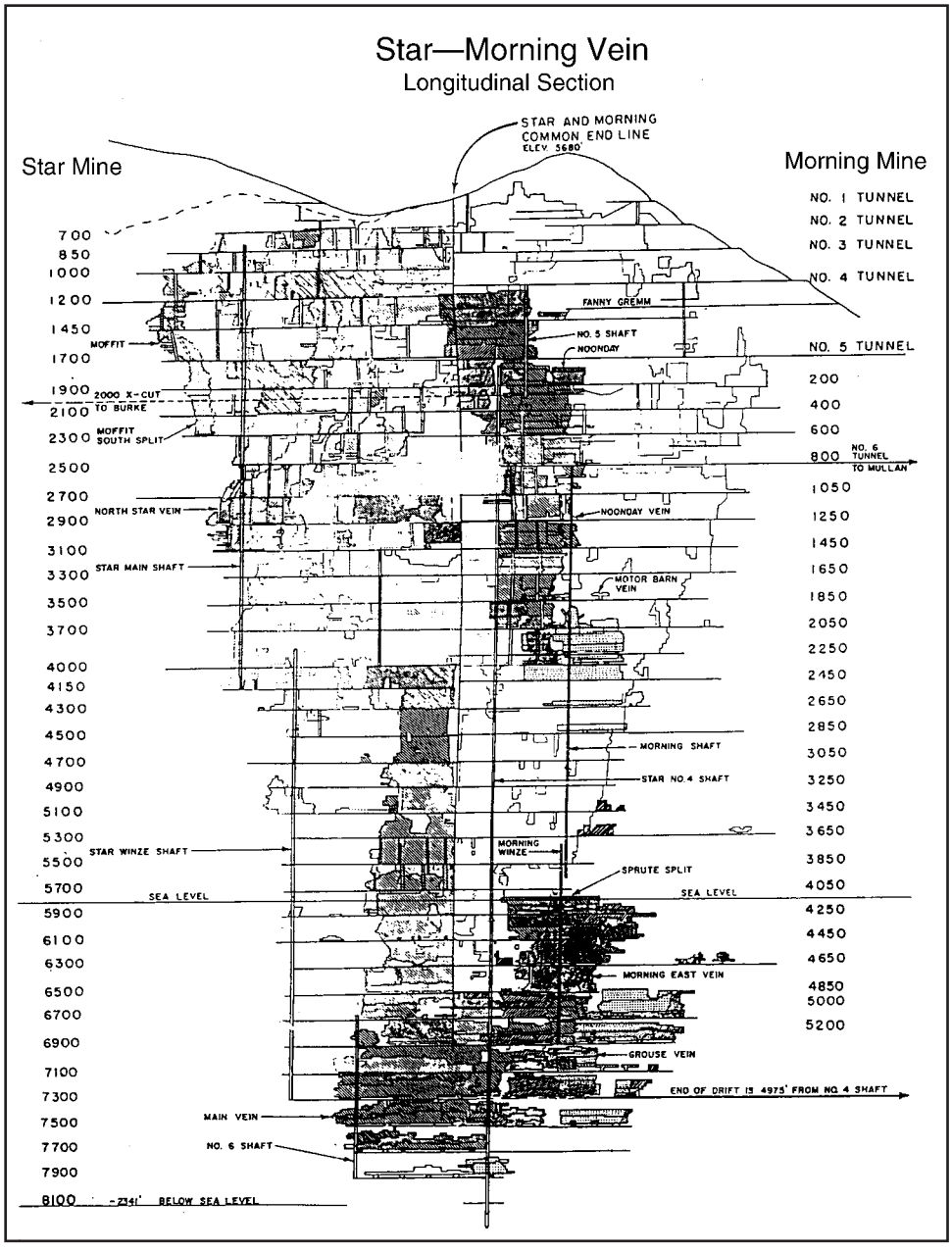


FIGURE 14.1 Longitudinal section of the Star-Morning Mine

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

service mining below the 7100 level. It was later deepened in 61 m intervals as mining was carried out down to the 8100 level, about 2,440 m in depth. The mine was shut down in 1982 due to low metal prices.

ROCKBURST INCIDENTS

Reported rockburst incidents at the Star date back to the early 1940s during mining below the 3700 level. These bursts were infrequent and did not cause much damage. They were mostly associated with pillars created by mining up under a level or up against a mined-out opening. As the bottom of the adjacent Morning workings (mined prior to 1950) was approached, the incidence and severity of bursting increased. While there were reports of rockbursting associated with mining at the Morning, very little of this information is available. However, there were some large damaging bursts in the 1940s associated with mining on their 4450 level (Star Mine 6100 level) that were picked up by seismographs and reported in local newspapers.

With the opening up of the Star 6900 level, severe bursting was encountered in raising up and driving first floors. The mining geometry was a long flat back as stopes were mined up at about the same elevation. A rockburst double fatality occurred in 1964 as first floors from the No. 4 and No. 8 Stopes approached each other, creating a small pillar. Similar conditions, although not as severe, were encountered on the 7100 level. By contrast, the 7300 level was opened up and mined with relatively little bursting until sill pillar mining was brought through to the level above. There was a delay of about 2 years in opening up the 7300 level, waiting for the No. 4 Shaft to be completed, but it is not clear that this was the reason for the reduced incidents of bursting. The mining geometry for stopes progressing up from the 7500 level was changed to a center, lead stope, pyramidlike geometry. Despite this change, a recurrence of heavy bursting was encountered and persisted. Of the seven reported bursts in 1974, five occurred on the 7500 level and resulted in a total of 2,000 t of damage. Stope destressing was initiated to deal with this problem. This was accomplished by drilling 4 m long holes, angled up at about 45°, along the stope walls on about a 2 m spacing after a cut was completed. These holes would be loaded and blasted after the stope was sand filled. By 1975, the number of bursts had increased to 21 with 16 induced by the mining on the 7500 level. Damage associated with bursting on the 7500 level totaled about 1,500 t. Unfortunately, first-floor mining in a stope on the Morning East Vein on the 6900 level resulted in another rockburst fatality. During the period 1973 to 1978, 152 significant bursts were recorded at the Star, whereas only 39 were recorded at Lucky Friday.

The Hecla Mining Company had concentrated their rockburst effort at the Lucky Friday Mine in the early 1970s, owing to a more serious problem, but in 1974 it became necessary to increase the effort at the Star. Hecla commissioned a study of bursting that indicated that the problem was concentrated in two areas—bursting accompanying the opening up and mining of the first floors, particularly in the small pillar between converging floors, and bursting in sill pillars. This pattern had developed after the mining on the Star side extended below the old Morning workings at about the 6700 level. A 24-geophone microseismic monitoring system, identical to the one at the Lucky Friday, was recommended and installed in 1975, and a rock mechanics engineer was hired. A destressing-in-advance-of-mining test was also recommended that would allow initial

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

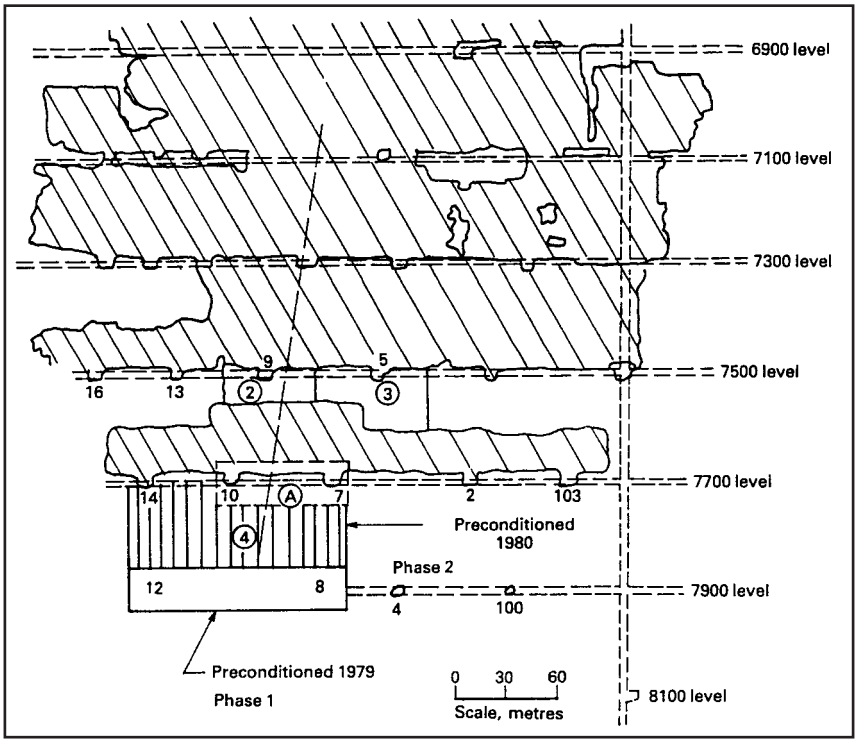


FIGURE 14.2 Mining geometry and stope preconditioning—Star Mine

mining to begin in a preconditioned zone and end in a preconditioned pillar. The concept was to drill and blast a large number of holes in order to fracture or precondition part of an entire stope block before initial development.

A test of preconditioning was carried out during 1975 and 1976 on the newly opened up 7700 level between the No. 7 and No. 10 crosscuts as a cooperative research program between Hecla and the U.S. Bureau of Mines. These stopes were selected because they were directly below the most burst-prone section of the 7500 level. About 35 long holes from each crosscut were ring drilled, then blasted, to fracture a zone some 76.2 m long by 12.2 m above and below the level. Some 1,372 m of 102 mm diameter drill holes were loaded with 4,082 kg of high explosive and blasted. There was no bursting during mining in the preconditioned zone, but bursting reappeared as mining continued above this zone. Of the 15 damaging bursts at the Star in the following year, more than half occurred in these two stopes above the destressed zone. During the next year, 20 of the damaging bursts at the Star occurred in these stopes. It was then decided to expand the preconditioning on the 7900 level to two entire stopes, as shown in Figures 14.2 and 14.3.

This preconditioning was carried out in two phases. The initial phase was to ring drill and blast holes drilled from each crosscut. The second phase of up-hole destressing was carried out after the second floor had been mined. This phase was finally completed in December 1980, and mining resumed in January 1981. By the end of 1981 the mining had almost reached the upper limit of the preconditioned zone in the 7900 No. 8 Stope.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

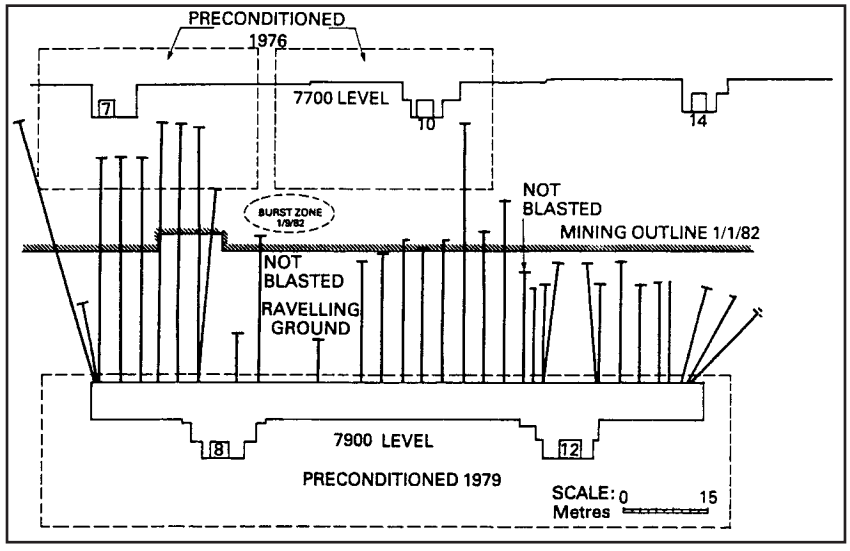


FIGURE 14.3 Phase 2 preconditioning, 7900 level—Star Mine

Only moderate seismic activity and no bursting had accompanied this mining. In January of 1982 the microseismic activity above the No. 8 stope increased and larger events began to register on the seismograph. On January 9 a 1.9 M_l burst occurred with a raise-up round and knocked out some 12 sets of timber, as well as displacing about 250 t of rock in this stope. Destressing down from the 7700 level No. 7 crosscut was initiated and was to be completed prior to resuming mining in this stope. However, due to low metal prices, the 7900 level stopes were shut down and never completed. The mine shut down later in 1982, and the lower levels never reopened.

DISCUSSION

Rockbursting increased significantly when the 6900 level was mined on the Star side below the mined-out Morning workings. While it appeared that the earlier mining on the Morning side had a destressing effect on later mining on the Star side, the combination of changed geological conditions and increased production owing to the No. 4 Shaft was not considered. In addition to the first-floor and sill-pillar bursting, a number of large bursts, up to 2.7 M_l, equal to the largest magnitude burst ever recorded at the Star, were located about 100 m out in the hanging wall. These bursts resulted in widespread shake-down damage and appeared to be occurring at about sill pillar elevations on a continuation of the Grouse Vein. An exploration drift out into the hanging wall confirmed the continuation of the Grouse Vein and a few stopes were developed and mined.

It was not uncommon for a large burst to be located up in an old, mined-out area of the Star. When old stope maps were checked, it was found that a waste pillar was left that would perhaps have burst years later. These pillars were completely surrounded by tailings backfill and confined by high fill pressure. Instruments placed in the backfill at the Star measured fill pressures up to about 13 Mpa and stope closures up to 0.5 m.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

It was also noted that though the Lower Revett Formation was very burst-prone, stopes located along thin-bedded sections of this formation were much less burst prone than stopes located in thick-bedded sections.

REFERENCES

Blake, W. 1984. Rock preconditioning as a seismic control measure in mines. In *Rockbursts and Seismicity in Mines, SAIMM Symposium.* Series No. 6. 225–229.

Karwoski, W.J., W.C. McLaughlin, and W. Blake. 1979. Rock preconditioning to prevent rock bursts—report on a field demonstration. *Report of Investigations 8381*. Washington, DC: U.S. Bureau of Mines.

Waddell, G.G. 1966. In-situ measurement of rock deformation in a vein-type deep mine. *Report of Investigations 6747.* Washington, DC: U.S. Bureau of Mines.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 15

Galena Mine Wallace, Idaho

SUMMARY

The Galena Mine started to experience rockbursts in 1956 when the top part of the Silver Vein was initially mined. Mining access was changed to prevent the formation of small pillars, and the mining geometry and sequence was changed to limit the size of sill pillars. With increased depth, sill pillars became the primary bursting problem. Destressing has been routinely carried out to control sill pillar bursting.

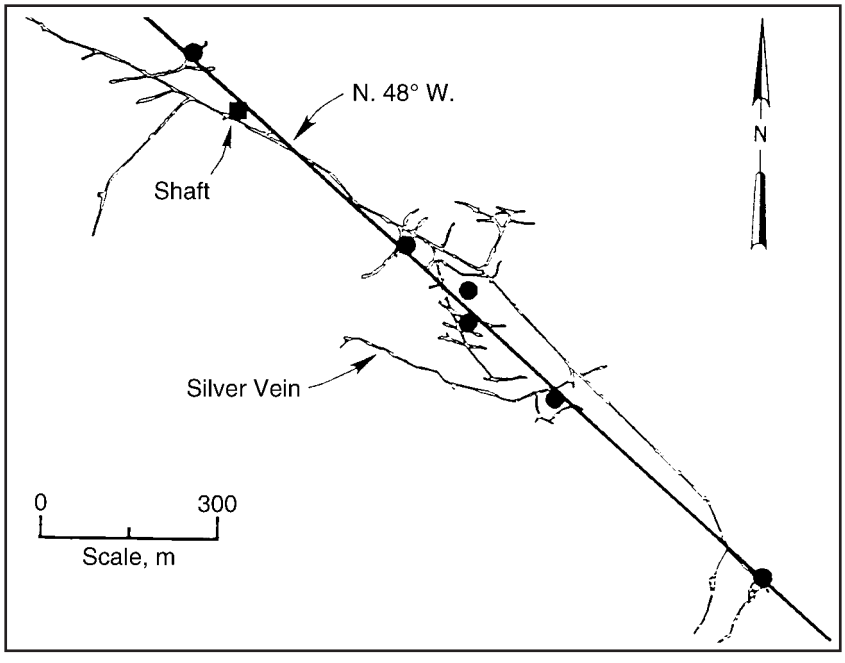
GEOLOGY AND MINING METHODS

The deep silver veins of the Galena Mine occur along a shear zone between two major faults. In contrast to the large single-vein systems at the Lucky Friday and Star mines, there are a very large number of small- to medium-sized veins all along the approximate 2 km length of the shear zone that runs through Galena. The majority of these veins are oriented normal to the shear zone, which strikes some N50° to 60°W and is parallel to the direction of maximum principal stress. Figure 15.1 is a plan of the 4300 level. The steeply dipping siderite veins, 1–2 m wide, contain argentiferous tetrahedrite and minor amounts of chalcopyrite. The wall rocks in the upper part of the Galena Mine are in the Wallace and St. Regis formations, whereas lead and silver veins below the 2200 level are primarily in the harder Revett quartzite. The strike of the bedding is generally parallel to the shear zone, and the beds dip steeply to the northeast.

Mining is carried out by overhand cut-and-fill using hydraulic sand fill. Initial upper level mining was accessed by the Callahan Shaft. The Galena Shaft was deepened in stages to the 5500 level. The No. 3 Shaft was also deepened in stages to the 5500 level. The 5500 level, some 1,677 m below the shaft collar but up to 2,000 m below surface, is currently the deepest operating level.

In 1953 an exploration drift to the southeast of the Galena Shaft intersected the top of the Silver Vein. Mining on this vein began in 1955 on the 2400 level. A 6 m sill was left above the level, and flat-backed cut-and-fill mining progressed upwards from three raises. By the end of 1957, mining from the 2400, 2800, and 3000 levels had induced some 32 damaging rockbursts, the largest of which was felt on surface as a local earthquake.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.



Note: Rockburst-prone stopes on Silver Vein are omitted for clarity. Plan view shows 4300 level at a depth of 1.3 km.

FIGURE 15.1 N48°W trend of stopes (solid circles) experiencing rockbursting and other large seismic events

ROCKBURST INCIDENTS

Initial bursting at the Galena was a cause for concern. After studying the problem, and with the help of consultants, the mine concluded that it was due to a combination of mining and geological factors. The top of the Silver Vein has a triangular shape owing to the convergence of two faults, which appeared to control mineralization. A concentration of intersecting mining and development openings between these faults, as well as the small sill pillars left above the 2400 and 2600 levels formed numerous burst-prone structures. With hard, brittle quartzite wall rocks and high-stress conditions, rockbursts began to occur. To deal with bursting, the mine changed the mining sequence below the 3000 level to a reverse stair-stepped geometry in order to isolate pillars. The small sill pillars were mined through to the level above by end slicing. The practice of leaving a small pillar above a level had already been discontinued, and it was concluded that the present stope-bolting practices, sand-filling procedures, and blasting practices were appropriate and should continue.

By 1968, the initial 20-bursts-per-year incidence rate had decreased to less than 10 bursts per year as a result of the mining geometry changes. Meanwhile, exploration had found numerous additional veins and production was increased to about 590 t per day. Sill pillars were still the basis of the rockburst problem.

Joint research with U.S. Bureau of Mines determined that microseismic monitoring could delineate burst-prone mining structures, particularly sill pillars, and that pillar

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

destressing could be used to fracture a pillar and transfer the high stress to abutments. It should be pointed out that this research effort was primarily carried out on the Silver Vein, which is the most extensive vein at Galena. By 1970, an automatic 7-channel seismic monitoring system was developed and installed. Since that time, both microseismic and seismic monitoring systems have been in continuous operation, with numerous equipment and data analysis upgrades.

The majority of seismic activity induced by mining, including damaging bursts, occurs within about 45 minutes of normal blasting activity. Rockbursts greater than 1.0 M_l occur about two times per month.

DISCUSSION

When it was recognized in the late 1950s that small pillars between mining and development openings resulted in rockbursts, mining geometry and the orientation of access openings were modified to minimize the formation of burst-prone structures. Developing a planned mining sequence to allow for better rockburst control on the Silver Vein resulted in a reverse, stair-stepped geometry starting on the east abutment. Microseismic monitoring and pillar destressing have been routinely used to minimize sill pillar bursting.

However, with increased depth, bursting has continued, and bursting out in the walls began to be associated with sill pillar geometry, particularly on smaller multiple-vein systems. Conventional sill pillar destressing is modified depending on the local geology and structure. Stope wall and back reinforcement consist of a combination of conventional rock bolts, grouted rebar-type bolts, splitset bolts, and mats or chain link mesh, as required. Timber and timber stulls are also used.

Stopes oriented perpendicular to the N45°W trend, the direction of maximum principal stress, are more prone to rockbursts, as indicated in Figure 15.1, than stopes oriented more parallel to this trend. However, the local geology and geologic structure have a major influence on the incidence of bursting induced by mining in any stope or vein system, particularly where there may be parallel or multiple veins.

Recently both overhand and underhand cut-and-fill mining began on a new vein on the 5500 level. This mining geometry is being carried out without creating pillars.

REFERENCES

Blake, W. 1972. Rockburst mechanics. *Quart. Colo. School Mines*. 67(1).

Swanson, P.L. 1995. Influence of mining-induced seismicity on potential for rock bursting. In *Proceedings: Mechanics and Mitigation of Violent Failures in Coal and Hard-Rock Mines,* Bureau of Mines Special Publication 01-95. 231–241.

Swanson, P.L., and C.D. Sines. 1991. Characteristics of mining-induced seismicity in a deep hard-rock mine. *Report of Investigations 9393.* Washington, DC: U.S. Bureau of Mines.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 16

Sunshine Mine Kellogg, Idaho

SUMMARY

The Sunshine Mine first experienced rockbursts in 1939 when mining on the Sunshine Vein entered the Upper Revett Formation. In 1941 some 27 rockbursts were reported including one which resulted in a double fatality. To deal with this problem, mining on the Sunshine Vein was reduced to a few widely spaced stopes and limited to a single shift. This successful mining strategy of limiting the number of stopes in burst-prone veins and single shifting has continued to date. Most veins at the Sunshine are less burst prone than at other mines in this district because they occur in more yielding wall rocks, they are oriented parallel to the direction of maximum stress, and they contain large waste inclusions. The most serious bursting problem at Sunshine has been small strain energy burst associated with driving raises.

GEOLOGY AND MINING METHODS

The mineralized zone at the Sunshine Mine occurs in a block of ground about 2.3 km vertical by 0.8 km wide and 3.2 km long. Production has come from some 30 siderite-quartz veins containing tetrahedrite-rich ore shoots. Most veins strike nearly east-west and dip steeply to the south, but minor mineralization occurs on northeast and west-northwest structures. The siderite-quartz veins pinch and swell along both strike and dip, but are generally less than 1.5 m in thickness. Figure 16.1 shows a long section view of the major ore zones at Sunshine.

The host rocks for the mineralization are the St. Regis and Revett formations of the Precambrian Belt Series. The St. Regis Formation consists of interbedded argillite and siltite with minor quartzite intervals. The upper part of the Revett Formation is characterized by quartzite with varied amounts of argillite and siltite. The beds strike about east-west and dip steeply to the south, generally more steeply than the veins. Mineralization is best developed in the folded north limb of the Big Creek anticline, which is intersected by a number of north-west and east-striking faults. Figure 16.2 is a plan view of the 3700 level showing the geology. The major principal stress at the Sunshine is horizontal and is also oriented about parallel to bedding and the strike of most of the veins.

Initial mining took place before 1900 and was accessed from surface tunnels. A modified square set with fill stoping was initially used, followed by cut-and-fill mining

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

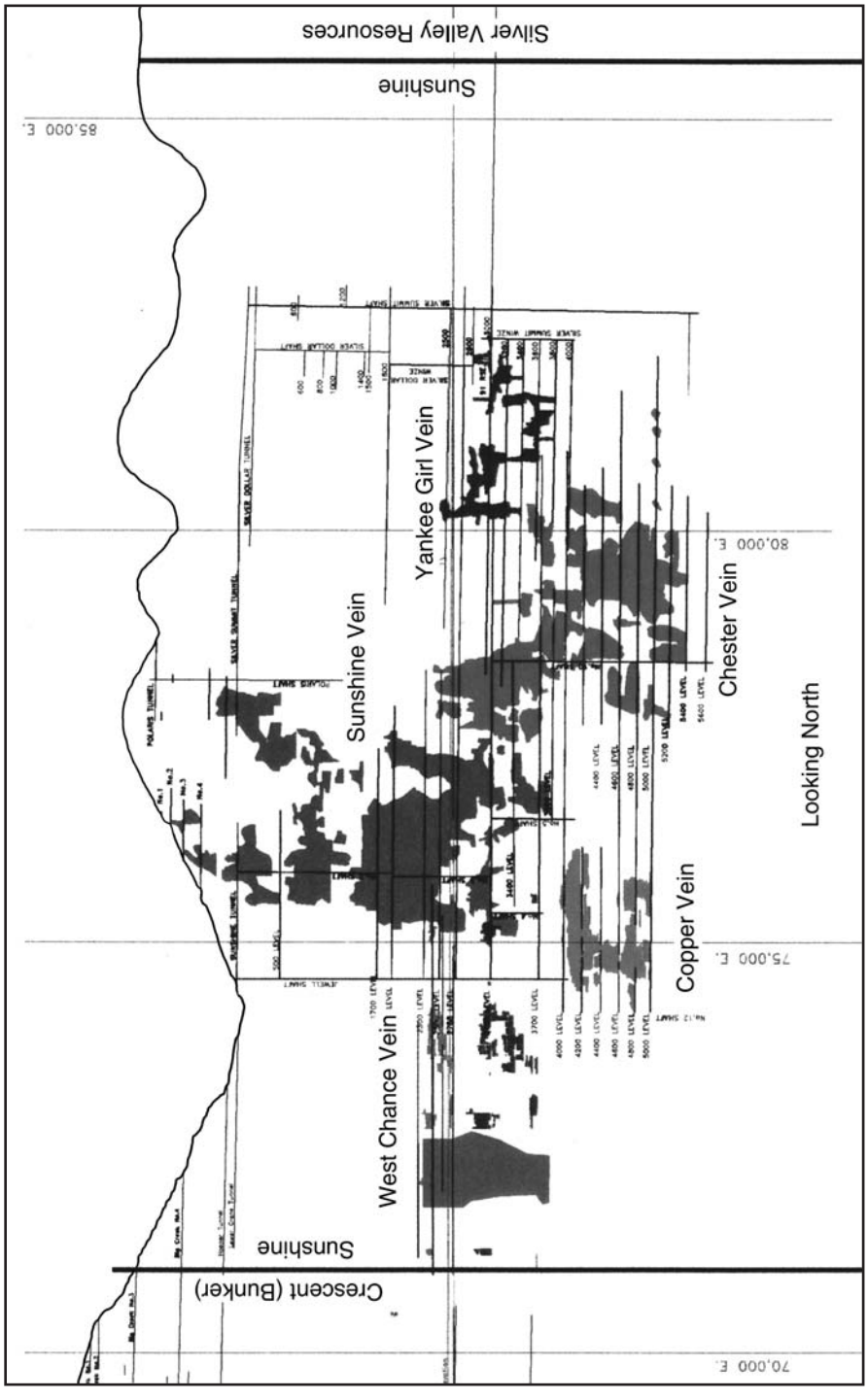


FIGURE 16.1 Sunshine Mine, longitudinal section

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

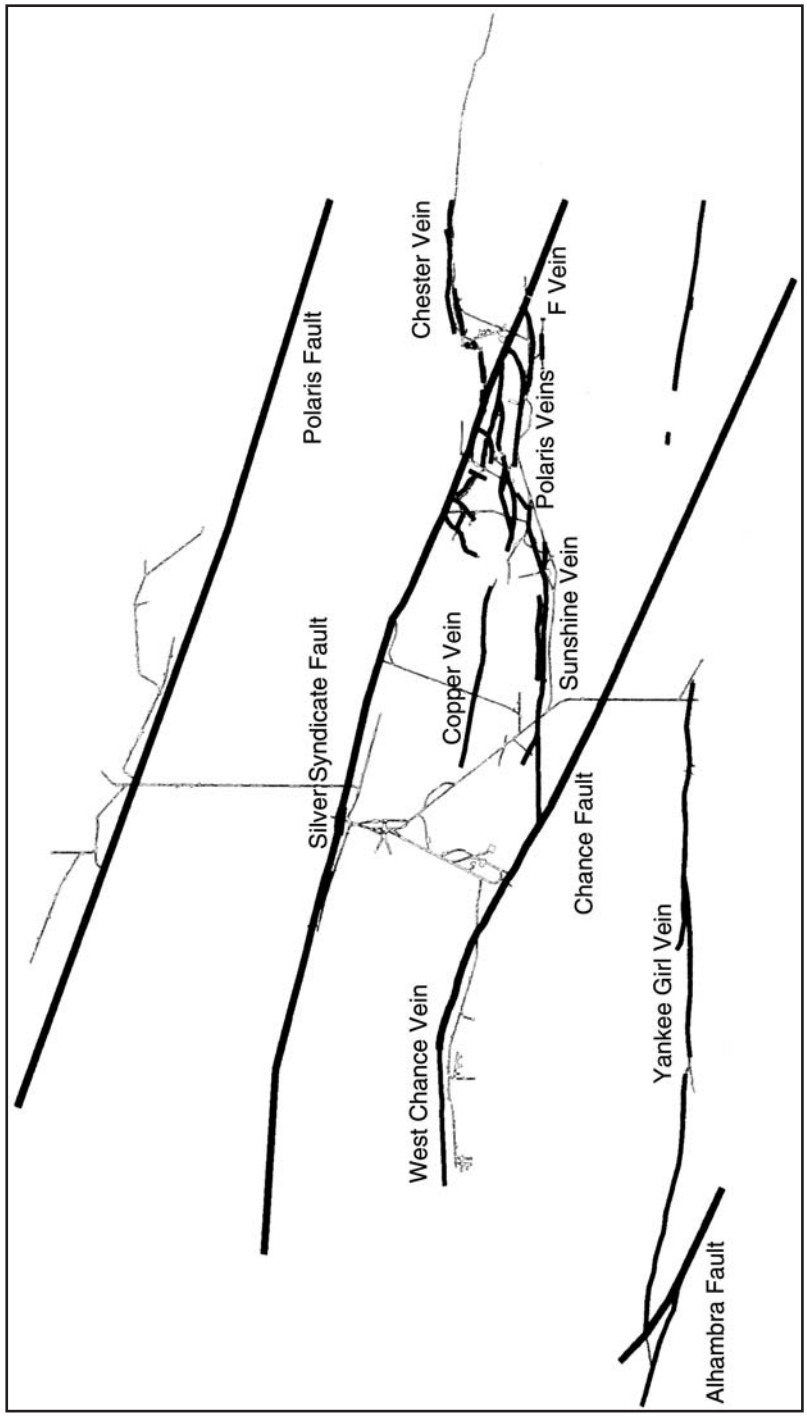


FIGURE 16.2 Sunshine Mine, 3700 level

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

with hydraulic sand fill. Recently, conventional cut-and-fill mining is being converted to mechanized cut-and-fill mining from local ramp access. The Jewell Shaft was sunk from surface and deepened to service mining on the Sunshine Vein. Mining below the 3700 level was serviced by a number of internal shafts as exploration discovered veins to the east and west of the main Sunshine Vein. The deepest level in the mine was the 5600 level, which is about 2,100 m below surface. Production at the mine was stopped in February 2001, and the mine was put on a standby basis.

ROCKBURST INCIDENTS

The first reported rockburst at the Sunshine Mine was on April 29, 1939, along the 2500 level drift in a waste section between the Sunshine "A" and "B" veins. This burst was felt on surface, and about 65 m of this access drift was badly damaged, along with damage reported along the 2700 level drift, and in some mining and access openings between the 2500 and 2700 levels. Crews were pulled from the work areas and kept on the levels until the mine quieted down. An investigation of this burst was carried out which revealed that the transition zone between the softer Lower St. Regis rocks and the much harder Upper Revett rocks started at about the 2500 level. The second burst occurred in July 1939, was more severe, and was located on the 2700 level of the Sunshine Vein near the bottom of the waste zone. During 1940 there were 10 bursts, 6 classified as heavy, along or just below this waste zone between the 2300 and 2900 levels. During 1941 some 27 bursts were reported, with 6 of these bursts occurring in the west part of the Sunshine Vein where no bursting had previously been reported. In June of 1941 a double fatality occurred on afternoon shift in the No. 1 stope in the B vein when a burst brought in the hanging wall. In September 1941, the U.S. Bureau of Mines visited the Sunshine Mine to test their recently developed microseismic monitoring equipment. Initial monitoring indicated that microseismic noise could be detected and that microseismic monitoring of mining at the Sunshine could be helpful. Plans to put together a monitoring system had to be deferred due to long delays in getting some of the equipment because of the war.

Only five bursts were reported in 1942, two on the east end and three on the west end of the Sunshine Vein. There was not another induced burst reported until 1949 when a strong shock was felt on surface but could not be located underground. There was no need for microseismic monitoring after 1942; hence, plans for monitoring were eliminated.

From 1942 until 1958 some 2,273,000 t were mined without a reported induced burst. However, incidents of inherent bursts during mining and development work were reported. A number of factors were cited that led to this decline in bursting. The bad geometry between the A and B branches of the Sunshine Vein disappeared, and vein mineralization became scattered below the 2700 level. The new orebodies mined to the east along the Syndicate Fault and on the Chester veins to the north were mostly in the softer St. Regis Formation. Mining was also reduced to a single shift as a result of manpower shortages due to the war. During this period the mining of a large number of stopes on a single vein changed to a few stopes being mined on numerous and spread-out veins. In addition, the host rock changed from hard to softer rocks, and the overall rate of extraction slowed down.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

Since 1958 rockbursting at Sunshine has continued on an intermittent basis. The major burst problem, from a hazard point of view, has not been large sill pillar bursts, but rather smaller strain energy bursts in development openings. All three rockburst fatalities since 1950 were the result of small inherent or strain energy-type bursts that occurred while driving raises. Sill pillar bursts still occur, a few annually greater than 2.0 M_l, with the largest to date 2.9 M_l, but they do not occur in every sill that is created as is the case in some of the other mines of the Coeur d'Alene Mining District. Rockburst incidents did increase again in 2000–20 bursts between 1.0 and 2.0 M_l, and 4 bursts greater than 2.0 M_l. This increase was attributed to increased production to offset lower silver prices as well as production being concentrated on a few veins with increased sill pillar extraction.

The mine installed a 24-geophone microseismic monitoring system in 1975, which was upgraded to a 64-geophone interactive system in 1982. A macroseismic system was installed in the early 1990s. Ground support over the years was increased to provide better containment of the damage caused by rockbursts. Conventional rock bolts and mats were replaced by a combination of grouted rebar bolts and splitsets, with chain link mesh along the back and down the walls to the floor.

The mining geometry or mining method was not changed because of bursting. However, stope mining would not be concentrated in burst-prone areas, and stopes in such areas would be single shifted. In addition, stress shadowing by stope sequencing in multiple-vein areas was carried out. Heavier ground support was also installed. While destressing was tried in both stopes and raises, it was never found to be successful enough to be used on a consistent basis.

DISCUSSION

Since the initial bursting on the Sunshine Vein in the early 1940s, the strategy to deal with this problem has been to avoid concentrating the mining in burst-prone veins and to slow down the extraction rate in such areas. In some cases, burst-prone sill pillars have been left unmined. This strategy has been very successful with respect to mining safely in burst-prone veins. The real hazard has been unexpected burst occurrences during the driving of raises. This problem is difficult to solve since there is generally very little geological information available regarding wall rock geology during stope development. The problem areas are unknown until a burst-prone bed, geologic structure, or contact are intersected by the development heading, and bursting occurs.

Bursting at the Sunshine was essentially confined to mining in the hard quartzites of the Upper Revett Formation. And even there, the quartzites of that formation, on the western edge of the original sedimentary basin, are not as pure or as strong as those found farther to the east, such as at the Star or Lucky Friday mines. In addition, the majority of veins at Sunshine are parallel to the maximum horizontal stress; hence, sill pillars are not as highly stressed as might be expected. And further, the aerial extent of most of the veins is not extensive and includes a number of waste zones; therefore, stress concentrations in sill pillars are reduced. For all of these reasons, the Sunshine Mine has had only an intermittent sill pillar burst problem rather than a continuous one. Large sill pillar bursts have only occurred when the vein was located in a band of hard quartzite beds in the center of a large mined-out area.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

REFERENCES

Wavra, C.S., W.D. Bond, and R.R. Reid. 1994. Evidence from the Sunshine Mine for dip-slip movement during Coeur d'Alene District mineralization. *Economic Geology*. 89(3):515–527.

Whyatt, J.K., T.J. Williams, and B.G. White. 2000. Ground conditions and the May 13, 1994, rockburst, Coeur d'Alene Mining District, Northern Idaho. In *Pacific Rocks 2000*. Edited by J. Girard, Lieberman, Breeds, and Doe. Rotterdam: A.A. Balkema.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

CHAPTER 17

Rockbursting at Kirkland Lake and Coeur d'Alene

SUMMARY

The mines at Kirkland Lake and in the Coeur d'Alene were the most burst prone of any mining districts in North America. Mining at Kirkland Lake induced more and larger rockbursts than resulted from mining similar tonnages in the Coeur d'Alene. In addition, almost twice as many fatalities were reported at Kirkland Lake. Tables 17.1 and 17.2 list the mining and rockburst data for Kirkland Lake and the Coeur d'Alene.

TABLE 17.1 Reported mining and rockbursts at Kirkland Lake

Mine	Mining Period*	Largest Burst	Number of Fatalities	Deepest Level	Total Tonnage
Sylvanite	1931–1962	2.0	0	4000	5.1×10^6
Wright-Hargreaves	1932–1964	4.2	7	8100	9.9×10^6
Lake Shore	1931–1965	4.4	21	8075	16.6×10^6
Teck-Hughes	1934–1939	3.0	7	6140	9.6×10^6
Kirkland Lake gold	1929–1959	2.5	2	6050	3.1×10^6
Macassa	1936–1999	3.8	6	7050	9.9×10^6
District			43		54.2×10^6

* Time of first reported burst occurrence to mine closure.

TABLE 17.2 Reported mining and rockbursts at Coeur d'Alene

Mine	Mining Period	Largest Burst	Number of Fatalities	Deepest Level*	Total Tonnage
Crescent	1902–1981	2.0	0	4200	0.8×10^6
Sunshine	1885–2000	2.9	5	5600	13.0×10^6
Galena	1917–2002	3.2	8	5700	7.0×10^6
Coeur	1975–1998	2.0	1	3700	3.5×10^6
Lucky Friday	1925–2002	4.1	4	6100	7.2×10^6
Star	1888–1982	2.7	4	8100	26.0×10^6
District			22		57.5×10^6

* Some mines were deeper owing to shafts placed along valley floor.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

DISCUSSION

While there are many differences between the geologic settings at Kirkland Lake and in the Coeur d'Alene, there are also many similarities. The gold-bearing veins of Kirkland Lake were narrow and steeply dipping, as are the lead-zinc-silver veins of the Coeur d'Alene. In general, the wall rocks at Kirkland Lake were more massive and competent than those found in the Coeur d'Alene. Host rocks for the veins at Kirkland Lake consisted of syenite porphyry, basic syenite, and tuff, whereas the wall rocks in the Coeur d'Alene are a sequence of quartzites and siltite-argillites. The primary difference from a geological point of view is that the mineralized veins at Kirkland Lake occur along a regional break or parallel fault system that extends along strike through all the mines. In the Coeur d'Alene each mine has its own vein systems and local geologic structures.

The mining method at Kirkland Lake started out primarily as shrinkage but was converted to some type of overhand cut-and-fill with depth and increasing bursting problems. The fill was either waste rock or surface sand. Only Macassa converted to a slurry paste fill during its final years of mining. In the Coeur d'Alene a number of early mining methods were converted to overhand cut-and-fill with slurry mill tailings as fill. The Lucky Friday converted to underhand mining with a slurry paste fill in 1985, and the Galena has begun conversion of one stope to underhand mining with paste fill in 2001.

An interesting similarity of both districts was that bursting started at around the 2,000 ft level and continued with depth. However, the lowest levels of almost all the mines, particularly the deepest mines, were not the most burst-prone levels. In almost every mine there is not a good correlation between increasing depth and increasing bursting past a certain depth.

Lake Shore was the most burst-prone mine at Kirkland Lake, with 26 bursts greater than 3.0 M_n, including 7 bursts greater than 4.0 M_n. The Lucky Friday is the most burst-prone mine in the Coeur d'Alene, with 27 bursts greater than 3.0 M_l, but only two of these were of 4.0 M_l or larger.

With the exception of the Lake Shore Mine, fatalities for individual mines in both districts are comparable. It should be pointed out that the first 20 fatalities at Kirkland Lake, the majority of which happened at Lake Shore, occurred between 1934 and 1939. This prompted the formation of a districtwide committee to study the causes and control of rockbursts. In addition, R.G.K. Morrison, from the Kolar Gold Field, India, was retained by the Ontario Mining Association to study rockbursting in Ontario, with emphasis on Kirkland Lake. As a result of these efforts, 20 fatalities were reported during the next 25 years of mining at Kirkland Lake. Similarly, cooperative rockburst research was initiated in the Coeur d'Alene by the U.S. Bureau of Mines in 1964, following a double fatality. Since that time, 16 rockburst fatalities have been reported during some 35 years of mining.

The 1984 rockburst sequence at the Falconbridge Mine in Sudbury resulted in four fatalities, as well as being the driving force for a 10-year government–industry rockburst research program in Ontario. Since the initiation of this program in 1985, there have been five rockburst fatalities in Ontario mines, with three occurring in Kirkland Lake.

Since 1985 there have been five rockburst fatalities in the Coeur d'Alene. These data suggest that despite all the research and rockburst control measures developed over the years, rockburst fatalities will continue to occur when mining is carried out on narrow veins in burst-prone ground.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

REFERENCES

McMahon, T. 1988. *Rockburst Research and the Coeur d'Alene District*. Information Circular 9186. Washington, DC: U.S. Bureau of Mines.

Morrison, R.G.K. 1942. Report on the rockburst situation in Ontario Mines. *Trans. CIM*. 43:225–272.

Whyatt, J., W. Blake, T. Williams, and B. White. 2002. *Sixty Years of Rockbursting in the Coeur d'Alene Mining District of Northern Idaho, USA: Lessons Learned and Remaining Issues.* Paper presented at 2002 Annual Meeting, SME, Phoenix, AZ.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

Index

Note: *f* indicates figure; *t* indicates table.

A

Aerial extent of mining 11
American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc. 2

B

Brunswick Mine (New Brunswick) 15
  changes in mining methods 15–16, 16–18
  geology 15
  ground support 18
  long section 16, 17*f*
  microseismic monitoring 16–18
  overhand cut-and-fill mining 15–16
  pillars and rockburst problems 16
  primary-secondary open stoping 15–16
  pyramidal pillarless mining 15–18
  stress shadowing 18
Bumps 2, 3
Burke, Idaho. *See* Star Mine (Idaho)

C

Campbell Mine (Ontario) 83
  A (and other) zone rockbursts 86
  F zone rockburst sequence 84–86, 85*f*
  geology 83
  microseismic monitoring 84, 87
  mining methods 83–84
  pillar bursts 84–86
  remediation by using larger pillars 84–87
Canada Center for Mineral and Energy Technology (CANMET) 50
Coeur d'Alene District, Idaho 114. *See also* Galena Mine (Idaho), Lucky Friday Mine (Idaho), Star Mine (Idaho), Sunshine Mine (Idaho)
  mining and rockburst data 113*t*
Copper Cliff North Mine (Ontario) 71, 75
  geology 71, 72*f*, 74*f*, 75
  mining methods 71–73
  rockburst incidents 73
Creighton Mine (Ontario) 65
  geology 65, 66*f*
  mining methods 65–66
  rockburst control measures 69–70
  rockburst incidents 67–69, 68*f*

D

Depth 11
Destressing 9
  Galena Mine 104–105
  Star Mine 99–101, 100*f*, 101*f*

E

Elliot Lake, Ontario. *See* Quirke Mine (Ontario)

F

Falconbridge No. 5 Shaft Mine (Ontario) 53, 57, 63, 114
  early rockburst activity 54
  geology 53
  major rockburst sequence (1984) 54–56, 55*f*, 56*f*
  mining methods 53–54
  post-closing seismic activity 56
Fatalities 114
Fault-slip bursts 5–6
  Lake Shore Mine 23–27, 24*f*, 26*f*

G

Galena Mine (Idaho) 103
  geology 103, 104*f*
  microseismic monitoring 104–105
  mining methods 103, 114
  pillar destressing 104–105
  remediation measures 105
  rockburst incidents 104–105
Ground support
  Brunswick Mine 18
  Sunshine Mine 111

I

Idaho. *See* Galena Mine (Idaho), Lucky Friday Mine (Idaho), Star Mine (Idaho), Sunshine Mine (Idaho)
Induced bursts 2
Inherent bursts 2

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

K
Kellogg, Idaho. *See* Sunshine Mine (Idaho)
Kirkland Lake District, Ontario 114. *See also* Lake Shore Mine (Ontario), Macassa Mine (Ontario), Teck-Hughes Mine (Ontario), Wright-Hargreaves Mine (Ontario)
  mining and rockburst data 113*t*
Knocks 3

L
Lake Shore Mine (Ontario) 19, 27–28, 35, 39–40
  fault-slip bursts 23–27, 24*f*, 26*f*
  geology 19–21, 20*f*
  geology at depth 35, 36*f*
  mining methods 21
  mining methods at depth 37
  as most burst-prone at Kirkland Lake 114
  pillar bursts 22–23
  rockburst history at depth 37–39, 38*f*
  strain bursts 22, 39
Lucky Friday Mine (Idaho) 89
  geology 89
  mining methods 89–90
  as most burst-prone at Coeur d'Alene 114
  rockburst incidents 90–92, 91*f*
  rockbursts with underhand mining 92, 93*f*
  underhand cut-and-fill mining 90–92, 94, 95*f*, 114

M
Macassa Mine (Ontario) 41, 50–51, 114
  April 1997 rockburst 46–50, 48*f*
  geology 41–42
  longitudinal section 42, 43*f*
  longitudinal section showing rockburst damage 44, 45*f*
  mining methods 42
  November 1993 rockburst 44–46, 47*f*
  pillar bursts 44–50, 45*f*
  remediation measures 50–51
  rockburst history 44–50
  seismic systems 50–51
Metamorphosed rocks 7
Mine stiffness 7
Modeling
  back analysis of Lake Shore Mine rockbursts 25–27
  chain reaction of pillar failures (Campbell Mine) 86–87
Monitoring 9–11
  micro- and macroseismic at Sunshine Mine 111
  microseismic (Brunswick Mine) 16
  microseismic (Campbell Mine) 84, 87
  microseismic (Galena Mine) 104–105
  microseismic (Quirke Mine) 79
Morrison, R.G.K. 83–84, 114
Mullan, Idaho. *See* Lucky Friday Mine (Idaho)

N
New Brunswick. *See* Brunswick Mine (New Brunswick)

O
Onaping, Ontario. *See* Strathcona Mine (Ontario)
Ontario. *See* Campbell Mine (Ontario), Creighton Mine (Ontario), Falconbridge No. 5 Shaft Mine (Ontario), Lake Shore Mine (Ontario), Macassa Mine (Ontario), Quirke Mine (Ontario), Strathcona Mine (Ontario), Teck-Hughes Mine (Ontario), Wright-Hargreaves Mine (Ontario)

P
Pillar bursts 4*f*, 5
  Lake Shore Mine 22–23
  Macassa Mine 44–50, 45*f*
  Quirke Mine 79–81
  Teck-Hughes Mine 30–32, 31*f*
  Wright-Hargreaves Mine 32–33, 33*f*, 39
Pillars
  avoiding burst-prone 8
  bursts 4*f*, 5
  destressing 9
  and rockburst problems at Brunswick Mine (New Brunswick) 16
  stiffness 7
  violent vs. controlled failure 7

Q
Quartzites 7
Quirke Mine (Ontario) 77
  geology 77
  microseismic system 79
  mining methods 77
  pillar bursts 79–81
  rockburst history 77–79, 78*f*

R
Rate of production 11
Red Lake, Ontario. *See* Campbell Mine (Ontario)
Rock
  brittle and elastic 11
  metamorphosed 7
  quartzites 7
Rockbursts
  and aerial extent of mining 11
  and brittle, elastic rock 11
  and bumps 2, 3
  causes 7
  controlled failure 7
  defined 2
  and depth 11
  and destressing 9
  and extraction rate 8
  fault-slip bursts 5–6
  first occurrences 1
  frequency of small vs. large 11–12
  geologic considerations 8

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

geologic settings favorable for 7
geometric considerations 8
induced bursts 2
inherent bursts 2
knocks 3
large seismic events 2–3, 3*f*, 4*f*
largest (Solvay) 2
logarithmic relationship between magnitude and frequency of occurrence 12
mines that have experienced 1
minimizing 7–11
monitoring 9–11
multiple large 11–12
Ontario classification system 2
and peak particle velocity of seismic wave 12–13
pillar bursts 4*f*, 5
and rate of production 11
reinforcement to contain 8–9, 10*f*
rules for controlling 13
seismographic records 3, 6*f*
small seismic events 3
strain bursts 3
violent failure 7
warning 13
worst disaster (Springhill) 2
yielding support 9, 10*f*

S

Star Mine (Idaho) 97, 101–102
geology 97, 98*f*
mining methods 97–99
rockburst incidents 99–101
stope destressing and preconditioning 99–101, 100*f*, 101*f*
Strain bursts 3
Lake Shore Mine 22, 39
Sunshine Mine 111
Wright-Hargreaves Mine 32–33, 33*f*
Strathcona Mine (Ontario) 59
Deep Copper Zone 61
geology 59
Main Ore Zone 59–61
main sill pillar area 59, 60*f*
mining methods 59–61
rockbursts in deep copper 62–63
rockbursts in main sill 61–63
Sudbury and Sudbury Basin, Ontario. *See* Copper Cliff North Mine (Ontario), Creighton Mine (Ontario), Strathcona Mine (Ontario)
Sunshine Mine (Idaho) 107, 111
geology 107, 108*f*, 109*f*
ground support 111
micro- and macroseismic monitoring 111
mining methods 107–110
rockburst incidents 110–111
strain bursts 111

T

Teck-Hughes Mine (Ontario) 29, 34
geology 29
mining layout 30, 31*f*
mining methods 30
pillar bursts 30–32, 31*f*
rockburst history 30–32, 31*f*

U

U.S. Bureau of Mines 114

W

Wallace, Idaho. *See* Galena Mine (Idaho)
Wright-Hargreaves Mine (Ontario) 29, 34, 35, 39
geology 29
geology at depth 35, 36*f*
mining methods 30
mining methods at depth 37
pillar bursts 32–33, 33*f*, 39
rockburst history 32–33, 33*f*
rockburst history at depth 37–39, 38*f*
strain bursts 32–33, 33*f*

Y

Yielding support 9, 10*f*

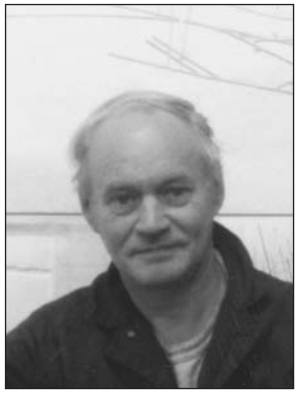
© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.

About the Authors



Wilson Blake was born in San Francisco, California. He graduated from the University of California, Berkeley, with a B.A. in geology in 1957, and after military service, he returned to Berkeley to earn an M.Sc. in engineering science in 1962. He joined the Denver Mining Research Center of the United States Bureau of Mines in 1965, while he worked toward his Ph.D. in mining engineering at the Colorado School of Mines. After receiving his doctorate in 1971, he became Director of Mining Research for GECAMINES in Lubumbashi, Zaire, in 1972, and has been an international mining and geotechnical consultant since 1975. He has authored numerous technical articles, primarily on the topic of seismic monitoring for deep mines as well as methods of dealing with rockburst problems.



David G.F. Hedley was born in northern England. He graduated with his B.Sc. from the Royal School of Mines, Imperial College, London, in 1961. He continued his studies at the University of Newcastle-upon-Tyne, winning his Ph.D. in rock mechanics in 1965. That year he immigrated to Canada and joined the Mining Research Centre of CANMET. In time he became senior research scientist. From 1990 until his retirement in 2001, he was a private consultant. He has authored many technical articles, mainly on the design of the Elliot Lake uranium mines and rockbursts in Ontario mines.

© 2003 by the Society for Mining, Metallurgy, and Exploration.
All rights reserved. Electronic edition published 2009.