ITASCA
Consulting Group, Inc.

28 October 1987

David Tiktinsky - SS623
U.S. Nuclear Regulatory Commission
Division of Waste Management
Washington, D.C. 20555

"NRC Technical Assistance
for Design Reviews"
Contract No. NRC-02-85-002
FIN D1016

Dear David:

Enclosed is Itasca's trip report for participation in the examination of deep mining conditions at the Strathcona Mine in Onaping, Ontario. Please call me if you have any questions.

Sincerely,

Roger D. Hart
Project Manager

cc: R. Ballard, Engineering Branch
Office of the Director, NMSS
E. Wiggins, Division of Contracts
DWM Document Control Room

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ITASCA TRIP REPORT

DATE: 20-22 October 1987

LOCATION: Strathcona Mine and the Mines Research Department
Falconbridge, Ltd., Onaping, Ontario

PURPOSE: To examine deep mining conditions at approximately 3000 feet depth at the Strathcona Mine in Onaping, Ontario. In particular, the Strathcona Mine has a history of rockbursting as well as ground control problems characteristic of deep mining and is similar in many respects to what is expected at the BWIP site.

ATTENDEES: M. Board (Itasca)
K. Wahi (Gram, Inc.)
J. Buckley, D. Tiktinsky (NRC)

PREPARED BY: M. Board and K. Wahi

SUMMARY

J. Buckley and D. Tiktinsky (of the NRC), M. Board (of Itasca), and K. Wahi (of GRAM, Inc.) traveled to Sudbury, Ontario, on 20 October and toured the Strathcona Mine of Falconbridge, Ltd. on 21 October. The Strathcona Mine, located outside Onaping, Ontario, is roughly twenty years old and nearing the end of its economic life. At present, the main mine sill pillar is being extracted by blasthole panel methods between the 2100 and 2500 levels. The mine produces roughly 12-15,000 tons per day with a working crew of about 250. Orebodies in this district are lenticular and formed by hydrothermal alteration of the host rock adjacent to a pluton structure. Average orebody width is 100 feet or greater, with a strike length of 1000-1500 feet. The orebody itself is a copper-nickel-iron sulfide. The host rock is either a brittle felsic gneiss or norite. The main sill stresses are estimated to be: σ_H = 80 MPa, $\sigma_H/\sigma_V \simeq 2.8$.

A typical extracting sequence involves driving overcut and undercut drifts through the orebody on approximately 60 feet horizontal and 80 feet vertical centers (Fig. 1). Vertical blastholes of 4"-6" in diameter are drilled from overcut to undercut using down-the-hole hammer drills. Initially, a drop raise is blasted at the end of the overcut to create breaking room for subsequent blasts. Then, rows of holes on about 4-foot spacing are shot (about 3-6 rows/blast) so that the stope retreats transversely across the orebody. Blasting is accomplished using ANFO with booster primers and primacord legs with non-electric initiation. The broken rock is mucked up below (in the undercut) using remote-controlled LHDs. The remaining stope is roughly 100-150 feet in length by 80 feet in height by 20-25 feet in width. When complete, the opening is backfilled with a weakly-cemented sandfill. The mining cycle continues with extraction of the pillars left between blasthole stopes.

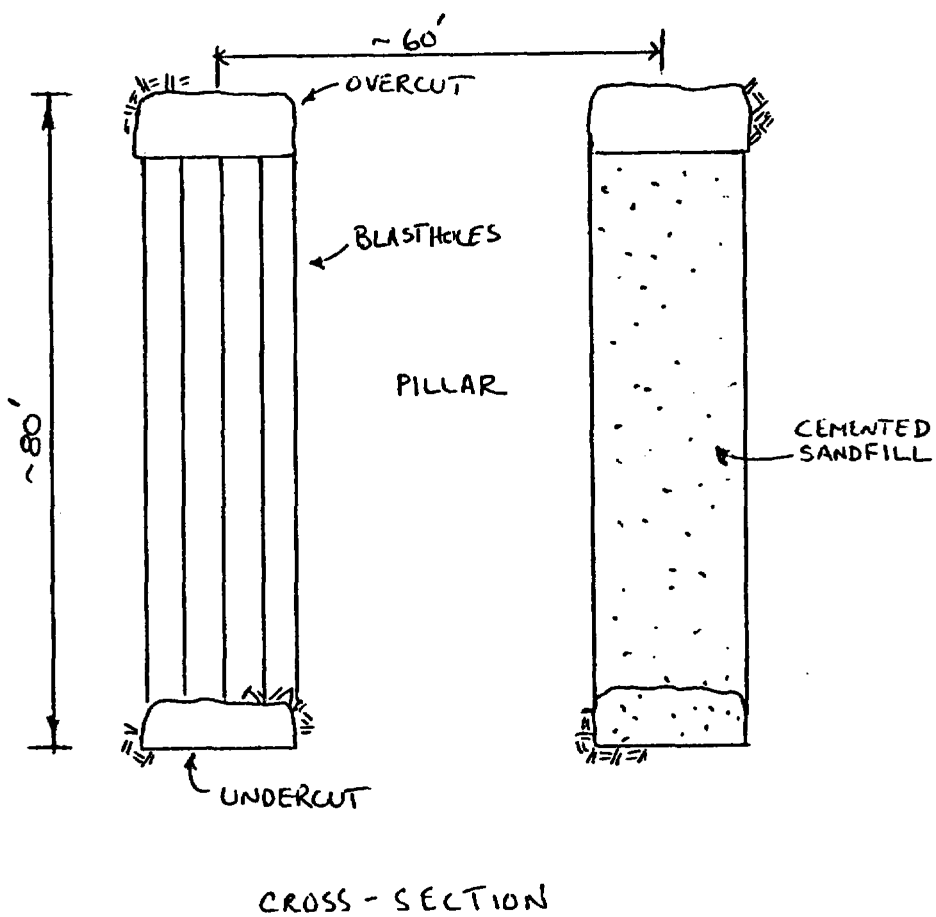

Fig. 1. Typical Blasthole Stoping Geometry at Strathcona Mine

Rockbursting can occur in either the orebody or host rock during mining. The predominant form of bursting is energy release due to unstable slip on existing fault planes or shear zones as a result of stress changes induced by mining. These events may register 3 or more in local Richter magnitude, sometimes resulting in great damage to underground workings. The attached paper describes a case history study of fault slip-induced rock bursting at the Strathcona mine, and analysis of the problem using discontinuum numerical models. Rockbursting may also occur as smaller events (<2 magnitude) in the face of advancing stope development headings. These events are probably related to the sharp stress concentration at the face and not induced slip on any particular feature.

Other ground control problems at the mine include caving in areas of high extraction and falls of ground from the back. The main types of ground support used includes point anchor and fully-grouted rockbolts (up to about 8 ft. in length) and cement-grouted cable anchors (up to 50-60 ft. in length). In all cases, wire mesh or expanded wire fabric is used to contain loose material. For severe problems in rockbursting ground, a passive support system known as wire rope lacing is used. Here, wire mesh is pinned to the rock using grouted rock bolts on 3- to 4-foot centers as per normal practice. Then, 3 foot-long smooth eyelet rebar are grouted into short holes in the walls and back using a diamond pattern on 3-foot centers. Continuous wire rope (slusher cable) is then wrapped from eyelet to eyelet in a triangular pattern. This system acts in a "soft" manner—i.e., under dynamic loads accompanying rockbursting, the support system flexes without failure, allowing loose material to be contained behind the wire mesh.

We initially visited the 2100 level where a cut-and-fill stope was in the process of extracting a small pod of ore by overhand mining (Fig. 2). When the sill pillar overhead was reduced to 40 feet, a 2 M_L rockburst occurred in the brittle wall rock adjacent to the pillar, resulting in several tons of rock displaced in both the stope and the above development level. At the present time, the raises into the stope are being rehabilitated so that equipment can be removed. The cause of this burst is unknown, and no precursory seismicity was noted. It was decided that the 2100 development would be wire rope laced and that the 40 foot sill pillar would be abandoned. The remainder of the orebody is to be taken by blasthole mining.

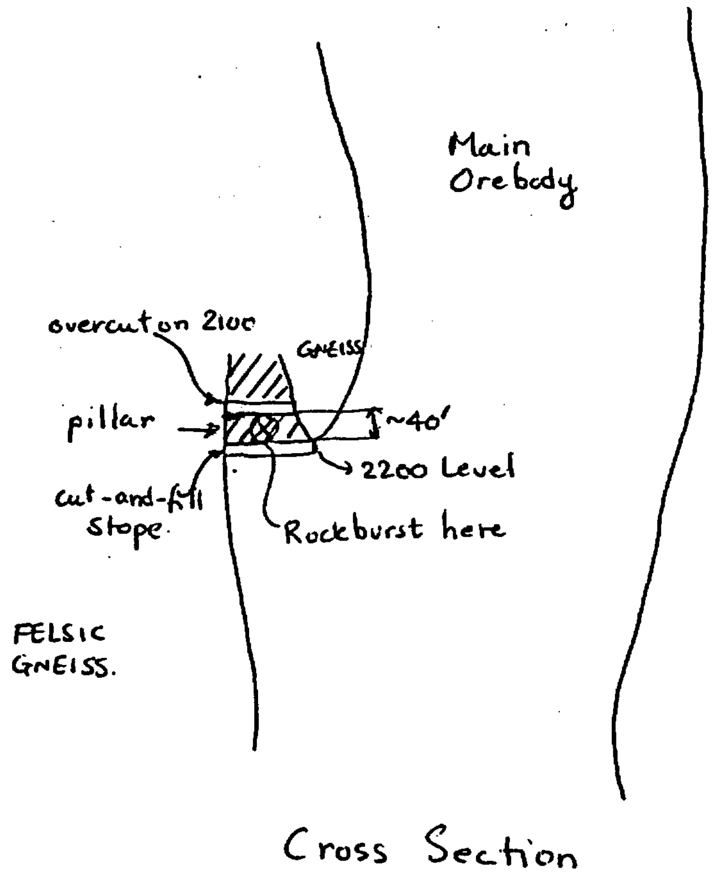

Fig. 2. Cross-section of Orebody Showing Bursting in Sill Pillar of Cut-and-Fill Stope

The site of three October-December 1985 rockbursts of 2-3 M_L was visited on the 2300-2500 levels. Here, wire rope lacing had been installed, in some places, prior to the rockbursting. It was seen that the effectiveness of the lacing was highly dependent upon the local geology. Where rock was very brittle, or where a fault crossed the opening, the lacing failed to support the back. Falconbridge staff feel that the rope lacing can withstand a peak particle velocity of over 6 in./sec., or a 3 M_L event at a distance of 180 ft., assuming reasonably uniform geology.*

*Glen Davidge, Chief Ground Control Engineer, personal communication.

Some research activities are currently underway in regard to rockburst control. Microseismic monitoring is being used in an attempt to identify rockburst-prone structures (e.g., faults, dikes) during or prior to excavation. The rate of activity is then being used as an indicator of stability. Presumably, the greater the rate of activity, the nearer the structure to instability. The microseismic systems have not, as yet, proven useful for prediction of bursting times.

The Queen's University of Kingston, Ontario is currently performing tomographic imaging of seismic velocity through a pillar in an attempt to determine velocity change prior to rockbursting as a precursory phenomenon. This research is in progress, with no results as yet.

Falconbridge and Itasca are performing joint research into the use of numerical models for identification of rockbursting problems. The discontinuum codes MUDEC and 3DEC are being used to analyze rockbursting as a problem in unstable fault slip.

The agreement obtained thus far between the model predictions of fault slip and microseismicity have been encouraging.* In the Spring of 1988, an experiment is planned for injection of water under pressure into the burst-prone fault structures. It is planned that the faults will be induced to slip non-violently, thereby releasing the energy in a stable fashion rather than building toward a large slip.

Conclusions

The trip was quite helpful in relating potential problems in ground control at the Hanford Site to those presently being experienced at the Strathcona Mine. The Hanford Site is known to have a high magnitude, highly deviatoric stress field ($\sigma_H \approx 70$ MPa, $\sigma_H/\sigma_V \approx 2.5 - 3$), both of which are conducive to rockbursting. Microseismicity at depth at the Hanford Site is apparently related to slip on existing features, either faults or interflow bedding features. The introduction of excavations will result in induced stresses which could result in slip on these features in much the same fashion as presently occurs at the Strathcona Mine. It is interesting to examine the consequences of this slip at Strathcona as it relates to performance at the Hanford Site. Listed below are several points of comparison.

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* See attached paper.

1. The stress states at the Hanford site and within the Strathcona main mine sill are of similar magnitude and ratio.

2. Slip of faults at Strathcona result in slip radii of greater than 50 meters (approximately 150 ft.) as evidenced by microseismicity. The fault slip resulting in rockburst has been seen to cause sudden ingress of water into excavations along the structure. There is a potential that slip radii of this magnitude for faulting at Hanford could interconnect the repository to adjacent interflow zones, resulting in an ingress of water. At Hanford, the interflows have a huge supply of pressurized water, thereby making ingress potentially dangerous.

3. It is not possible at Strathcona to tell, ahead of time, from geologic mapping, what structures appear to be most susceptible to slip. Sometimes, apparently insignificant features are burst-prone.

4. Wire rope lacing appears to do a good job of controlling fly rock from rockbursting, but it is very expensive to install.

Respectfully submitted,

Mark Board

Mark Board

attach.
MB:RLK

COST BREAK-OUT

Labor

M. Board	24 hrs @ $23.56/hr	$ 565.44
	TOTAL LABOR	$ 565.44

Actual Expenses

Travel

Airfare (Mpls-Sudbury-Mpls)	$ 386.05
Miscellaneous Travel Expenses (taxis)	24.00
Lodging (2 nights at $26.88/night)	53.76
Meals	42.00
Miscellaneous Expenses	
Telephone	5.50
TOTAL EXPENSES:	$ 511.31

Presented at the SME Annual Meeting
(Phoenix, January 1988), Geomechanics,
Session II

USE OF NUMERICAL MODELING IN MINE DESIGN AT FALCONBRIDGE, LTD.

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*currently of INCO Ltd., Sudbury, Ontario, Canada

INTRODUCTION

Falconbridge, Ltd. operates six mines (the Strathcona, Fraser, Lockerby, Onaping, East and Kidd Creek) in the Sudbury and Timmins districts of northern Ontario (Fig. 1). These mines vary in depth to approximately 1500m and experience a wide variety of ground conditions. Problems of greatest concern include rockbursting, backfill pillar stability, wall and pillar stability, and the use of cable anchors. A study was initiated approximately two years ago to identify how numerical models could aid in the examination and possible understanding or solution of these problems. This paper discusses the use of numerical models at Falconbridge and describes a case history of their use in examination of rockbursting at the Strathcona Mine.

SELECTION OF NUMERICAL MODELS

An examination of the ground control problems at Falconbridge mines was performed to identify the types of numerical models required. The problems of primary interest are:

(1) rockbursting as a result of fault slip and rock fracture;

(2) fill pillar stability and collapse;

(3) pillar wall stability and the effects of cable bolting; and

(4) access drift stability.

Analysis of these problems require the capability to simulate:

(1) discrete faults or fractures in the rock mass and slip and separation along these features;

(2) continuum non-linear material behavior with ability to model large displacements and possible strain-softening;

(3) two- and three-dimensional problem geometry;

(4) possible dynamic analysis;

(5) rock support;

(6) sequential mining;

(7) initial stress;

(8) ability to use varying material models;

(9) gravitational loads;

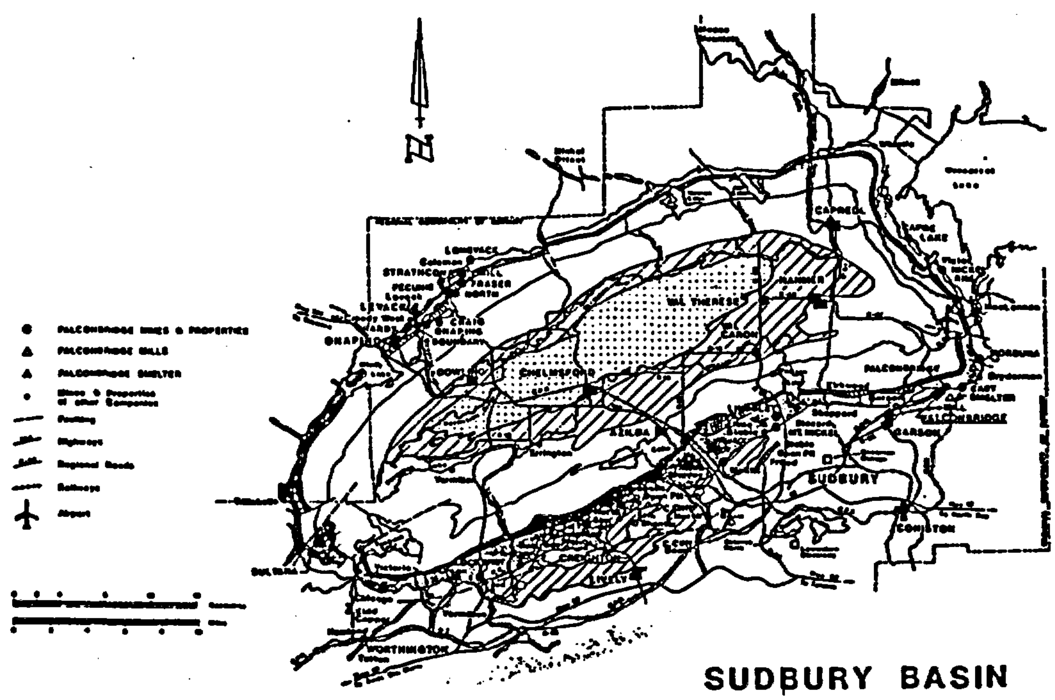


Fig. 1 Sudbury Basin Mines

(10) ability to model collapse mechanisms without numerical instability; and

(11) extensive graphical output of data.

An additional requirement was that the codes must operate on a personal computer without excessive run times.

The basic types of numerical models are reviewed in Fig. 2. Models may be grouped into two primary types: continuum or discontinuum. The continuum codes can be subdivided further into integral (boundary element) and differential (finite element and finite difference) methods. The boundary element methods (Crouch and Starfield, 1983) assume, in general, that the rock mass behaves elastically but have the advantage of ease in problem set-up and operation. Finite element and finite difference codes have the ability to examine highly non-linear continuum problems with complex geometries. The distinct element method (Cundall, 1980) can be used to examine problems in which discontinuum response cannot be analyzed as an equivalent continuum.

MODEL GROUPS

	CONTINUUM			DISCONTINUUM
	Integral	Differential		
		Implicit	Explicit	Explicit
ROCK MASS BEHAVIOR	Elastic Rock Mass Plastic Yield in Boundary Elements	Elastic/Plastic Rock Mass and Ubiquitous Jointing		Elastic/Plastic and Brittle Fracture of Blocks Slip, Dilation, and Separation of Joints
		must iterate to follow non-linear behavior	no iteration needed to follow non-linear behavior	
FAILURE DESCRIPTION	Confined Failure Region		Material Collapse	Material Collapse and Block Rotation and Separation

Fig. 2 Model Groups for Mining Analysis [Hart and Board, 1985]

It was determined that the broad range of potential problems presented the need for codes in all three basic model groups. Table I lists the problem requirements and the codes obtained to perform analyses in these areas. The two- and three-dimensional BESOL boundary element codes (Crouch Research, 1985) are intended for use in relatively simple analyses where the rock mass can be considered elastic or where any non-linear behavior is confined to the vein. In general, these codes are used for analysis of vein-type problems where there is no wide variation in thickness of the orebody.

For general two-dimensional problems in which the rock mass response is not controlled by isolated discontinuities, the explicit finite difference program FLAC [Itasca, 1987(a)] is used. This is a PC-based geomechanical-oriented code which includes, as major features:

(1) a variety of material models, including plasticity, ubiquitous joints and strain softening;

(2) automated mesh generation;

(3) large strain;

(4) support-structure interaction; and

(5) complete graphics representation of data.

Of particular importance in the FLAC (and distinct element) code(s) is the explicit or time-marching solution procedure (Cundall, 1976). Highly non-linear or large strain problems can be handled accurately without numerical instability because the solution procedure does not require the formulation and inversion of systems of equations.

Approximately 2,000 elements can be modeled on the standard 640K IBM-PC, and even larger problems can be modeled on UNIX-based PC processor boards, such as the Definicon Systems, Inc. 32-bit processor. With this board, which plugs into the standard IBM PC, as much as 16mb of RAM can be used, and execution speeds reportedly exceeding that of the VAX 11/780 are possible. It is thus possible to perform sophisticated modeling studies without the use of a mini- or mainframe computer.

The MUDEC (2-D) and 3DEC (3-D) codes [Itasca, 1986; 1987(b)] are distinct element models which are used to analyze problems in which fracturing controls the rock mass response. The distinct element method models the rock mass as a series of blocks which are separated by intervening joint planes. The strength of the joints is controlled by a Mohr-Coulomb slip condition but may be given more complex behavior such as a peak and residual strength envelope. The blocks represent intact material and may behave elastically or by some non-linear constitutive law. Other features of the programs include automatic block and mesh generators, dynamic analysis capabilities, rock support elements, and full screen and plotter graphics capabilities.

Table 1

CODES CHOSEN FOR VARIOUS PROBLEM AREAS

CODE	REF.	CODE TYPE	CODE USE	COMPUTER REQUIREMENTS
BESOL	[1]	2-D, 3-D Boundary Element	relatively simple geometry, elastic simulation	IBM PC or compatible, 20 Mb hard disk enhanced graphics, 640K RAM, 8087 math co-processor, digitizing table, optional HP pen-plotter
FLAC	[2]	2-D Explicit Finite Difference	general analysis of stoping geometries, pillar, backfill, stability, gravity problems, etc; used where continuum-type failure is evident; large strain capabilities	as above
MUDEC	[3]	2-D Distinct Element	general analysis of stoping geometries, pillar and fill stability; fault, joint control of failure response, dynamic analysis (blasting, rockburst), cable anchor modeling	as above
3DEC	[4]	3-D Distinct Element	detailed analysis of 3-D problems; can be used for analysis of fracture or continuum-type failure mechanisms; dynamic modeling, rock support	PC-based, as above; uses DSI* 32-bit processor board for calculations, enhanced graphics adaptor for display

[1] Crouch Research, 1985
[2] Itasca, 1987(a)
[3] Itasca, 1986
[4] Itasca, 1987(b)

*Definicon Systems, Incorporated

METHOD OF PROBLEM ANALYSIS

Before discussing a case example, the method by which a problem is approached and analysis performed is discussed. The logic for approaching a problem is given in Fig. 3. Initially, a problem must be identified and the potential behavioral mechanisms defined to the greatest extent possible. The assessment of failure mechanisms is generally based on two forms of field observation: visual and measured. Visual observation includes a detailed description of geologic structure and the relation of progressive damage or disturbance to the structure. Instrumentation measurements may include displacement or microseismic monitoring—but rarely more. As will be discussed in the following case example, field mapping and microseismicity have led to conjecture that rockbursting at the Strathcona Mine can be related to slip on a fault and dike structure.

Once a potential mechanism has been identified, the generally complex geologic structure must be idealized to a level which can be incorporated into the numerical model. This involves an assessment of the structural features which are of significance to the problem. In many cases, small-scale jointing and the possible non-linear behavior of the intact rock mass may be ignored because the primary response of interest is centered on the major structural features in the model. Consequently, the intact rock is considered elastic in nature. Other factors which need to be determined include the rock and discontinuity properties, the in-situ stress magnitude and orientation, whether a 2-D or 3-D analysis is warranted, and whether the analysis needs to be static or dynamic in nature. The idealization necessary is based, to a certain extent, on experience, but field observation and measurement must be used to the greatest extent practical. Rock mass properties are generally based, at least in the initial studies, on laboratory test data.

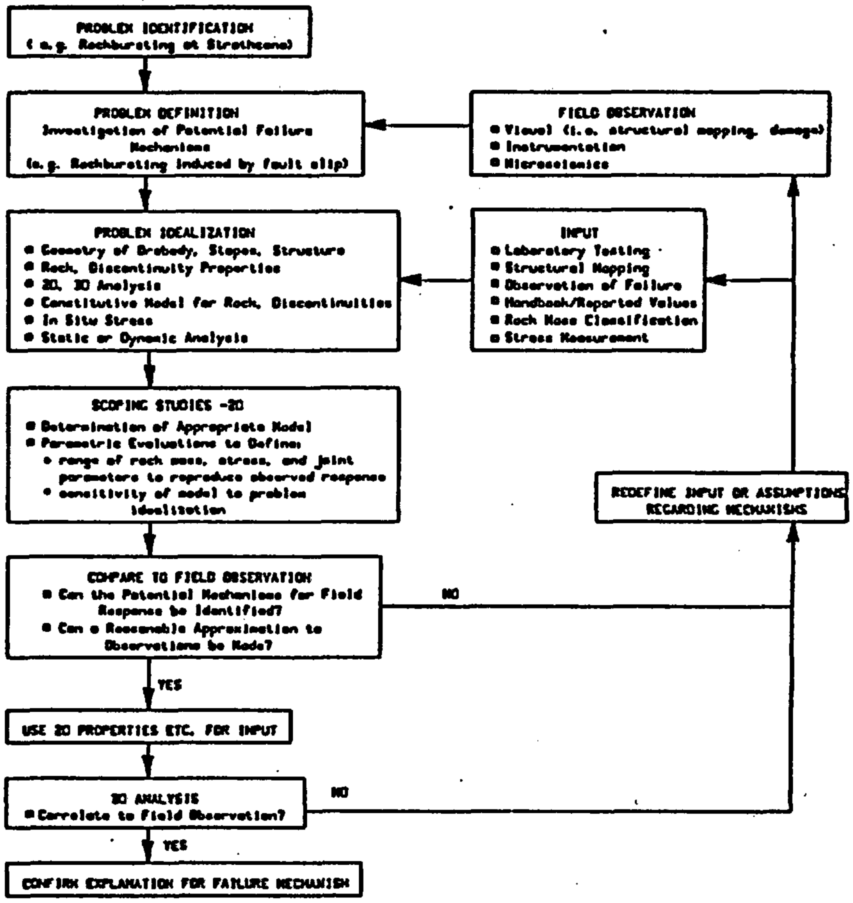


Fig. 3 Approach to Numerical Modeling of Ground Control Problems

Triaxial strength tests are used to define a Mohr-envelope and elastic properties. Tests on joints, if available, can be used to define stiffness, strength, and frictional properties. In the absence of field data, literature values for average properties can be used. In-situ stress measurements are generally available at the mine in question or within the district. Finally, a rock mass classification scheme may be used to yield some insight into the relationship of laboratory properties to field values (e.g. Hoek and Brown, 1980).

It is generally not time- or cost-effective to attempt immediately to perform a three-dimensional analysis of a complex mining problem. The modeling discussed here is all conducted on a personal computer and, therefore, cost is not an overwhelming consideration, but the memory and time required for three-dimensional runs can become restrictive quickly. The philosophy used here is to begin by conducting two-dimensional parameter studies. A typical two-dimensional scoping study may involve ten to twenty simulations of a mining sequence, whereas a three-dimensional study may involve less than five simulations.

It is the purpose of the two-dimensional scoping studies to define:

(1) the probable range of material properties;

(2) the structure and mechanisms which control the rock mass behavior; and

(3) the probable in-situ stress field orientation and magnitude.

The two-dimensional studies are an important precursor for three-dimensional analyses. The results of these studies provide the basis for continuing with the three-dimensional model. If a reasonable approximation to the observations cannot be made with the two-dimensional model, then it may be necessary to redefine the input parameters or assumptions. For example, if the principal stress orientation is defined for the model such that no fault slip is possible in the region in which slip was observed, then the stress orientation should be re-examined.

Once a consistent set of input conditions are defined, the three-dimensional analysis can then center on a general examination of proposed failure mechanisms and correlation of the model to field observations. Consequently, the three-dimensional analysis can confirm (or refute) the proposed mechanism for failure and provide insight into the factors controlling the failure.

At present, an extensive two-year program of code development and testing is being conducted. Six instrumented case histories from Falconbridge mines are being examined with the two- and three-dimensional codes as a means of code verification as well as providing data for further code development. The ultimate objective of these case studies is to build confidence in the models and modeling approach as greater experience is gained. As the studies continue, the codes will be applied to various planning or design projects as they occur.

A CASE HISTORY OF FAULT SLIP-INDUCED ROCKBURSTING AT THE STRATHCONA MINE

The Strathcona Mine lies on the northern rim of the Sudbury Basin; the main nickel orebody lies between 570m and 900m below the surface, dipping at 45-80° to the south. Previously, mining was conducted by mechanized cut-and-fill which has been replaced by a transverse blasthole panel method. Panels are about 10m to 15m wide by 25m high and of variable length, depending on the orebody width. Ore pillars of roughly 10m or more are left between weakly cemented sandfill pillars. Secondary or tertiary extraction involves mining of these pillars between one or two sandfill pillars, respectively.

This case study involves extraction of the primary stope panels in the main mine sill between 2250 and 2500 levels (Fig. 4). The following description of rockbursting during this extraction is adapted from Morrison (1987).

During the mining of the 25-200 panel, a great deal of seismic activity was reported and recorded by the mine-wide microseismic system. Two small rockbursts occurred, and the system identified the Main Dike (particularly in the footwall of the orebody) as the source of most of the activity during this period. Automatic plotting of event locations on level plans and sections was introduced in September 1985, and blasting of the 23-200 panel began in October 1985. The drop raise, 3mx3m, (Fig. 5) in this panel had been previously broken through to allow ramp development waste from the 2250 level to be added to the cemented tailings in the 25-200 panel.

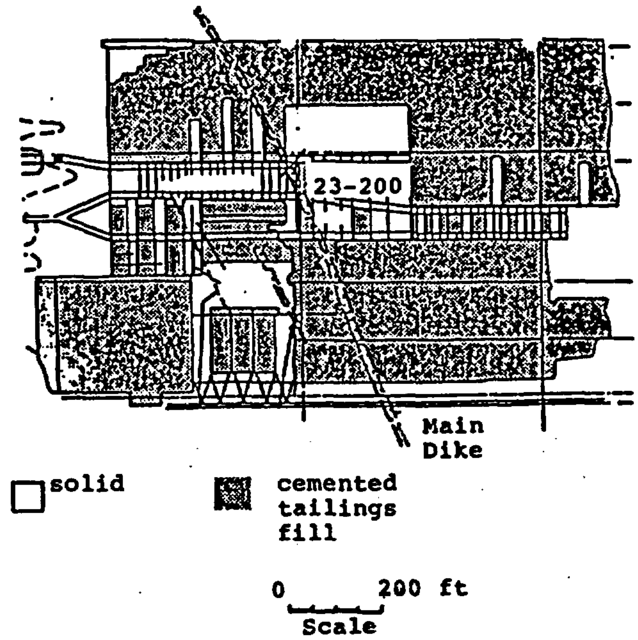


Fig. 4 Longitudinal Section of the Strathcona Mine Showing the Location of the Main Mine Sill, the 200 Panels, and the Main Dike Structure [Morrison, 1987]

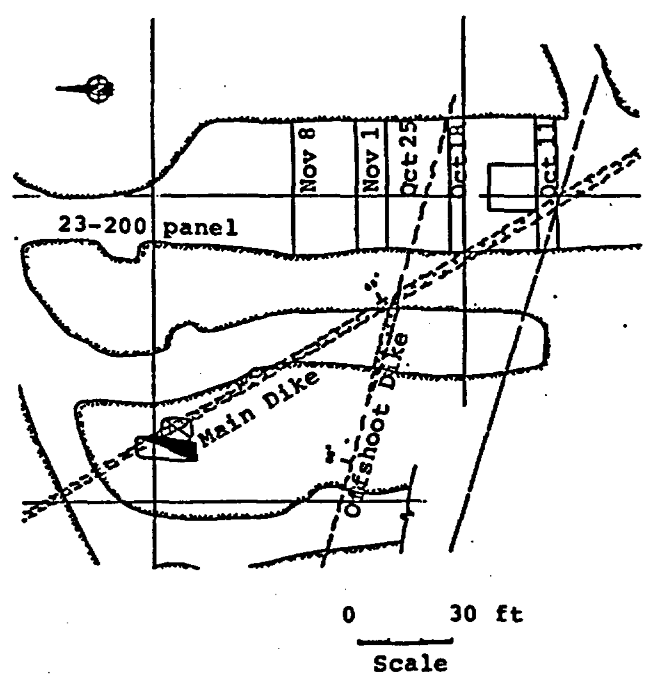


Fig. 5 Plan View of 23-200 Undercut Level Showing Blast Locations

The first two production blasts took place on 12 and 19 October at about 1:00 a.m. and were intended to open up the raise the full 14m width in preparation for retreat blasting toward the footwall access. The activity immediately following both blasts was expected to be on the dike, but most of the activity appeared on the 2250 level plan, forming a linear trend southeast of the stope striking 75° west of north. The trend after the first blast was noticeable, but after the second, larger, blast, it was quite pronounced (Fig. 6). On subsequent blasts (November 1, November 9), the trend did not appear, and the familiar pattern of activity around the open stope and along the dike in the footwall returned. A few large blocks ($30m^3$) were displaced from the western wall of the stope around the dike after some of these blasts.

At approximately 2:30 a.m. on November 9, some 90 minutes after the blast, a major rockburst occurred, registering $M_n = 2.2$. The blast appeared to be located in the footwall of the 220 panel. The damage was restricted to the 23+40 ramp elevation and in the open stope at least one very large block ($150m^3$) had been displaced from the west wall.

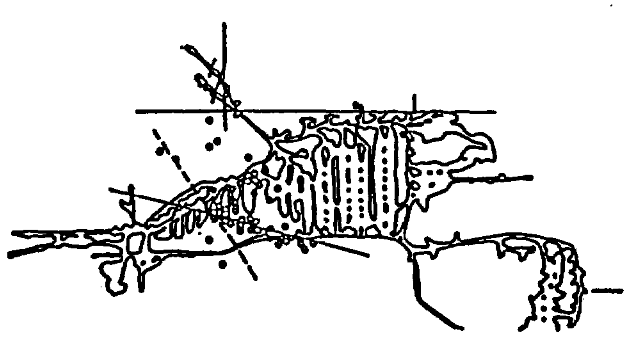
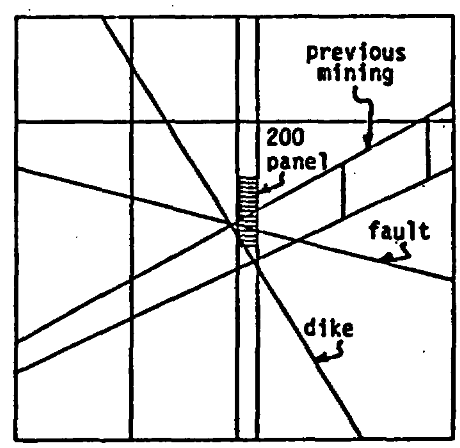


Fig. 6 Plan View of the 2250 Level Showing Location of Dike (dashed line) and Inferred Fault Structure (solid line) [Note: microseismicity aligns with the fault structure in the hangingwall of the orebody (Morrison, 1987)]

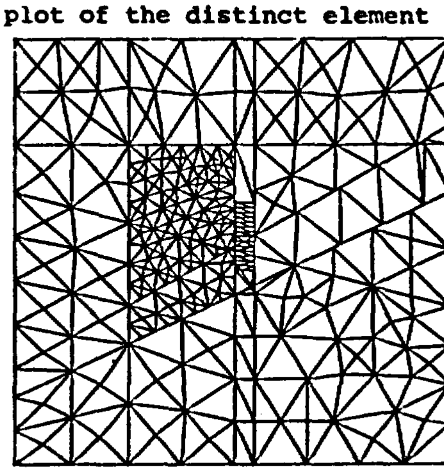
Two-Dimensional Analysis

As the remainder of the main sill is to be extracted in a similar fashion, a need exists to understand the mechanism by which these rockbursts occur and to assess the potential for further activity of this nature. A two-dimensional parameter study of this problem was conducted using the Micro Universal Distinct Element Code, MUDEC. The distinct element method was chosen because the rockburst behavior is apparently controlled by the fault and dike structures. A planar section through the 200 panel was chosen to construct the model for ease of comparison of microseismic data to model output (Fig. 7). Problem boundaries were chosen at 300m x 300m, resulting in a system of 35 blocks which were subdivided into 465 finite difference zones.

The input properties for the orebody and the host rock consisting of felsic gneiss and norite, as well as the in-situ stresses, are given in Table II (Daley, 1985).



(a) plot of the distinct element blocks



(b) internal block discretization into finite difference zones (The small central blocks represent the panel blasts to be removed.)

Fig. 7 Two-Dimensional Idealization of the 2250 Level, 200 Panel

Table 2

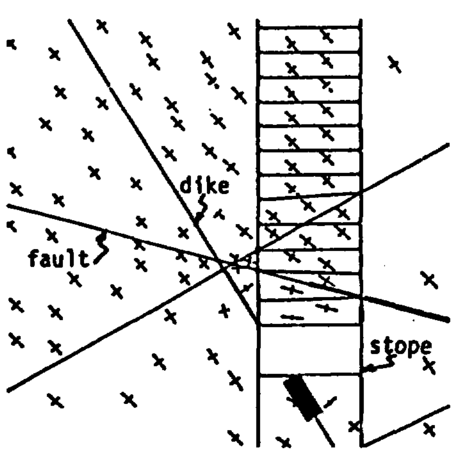
CASE STUDY INPUT PROPERTIES

UNIT	SHEAR MODULUS (GPa)	BULK MODULUS (GPa)
ore	14.2	19.0
felsic gneiss	24.3	47.2
norite	17.5	24.8

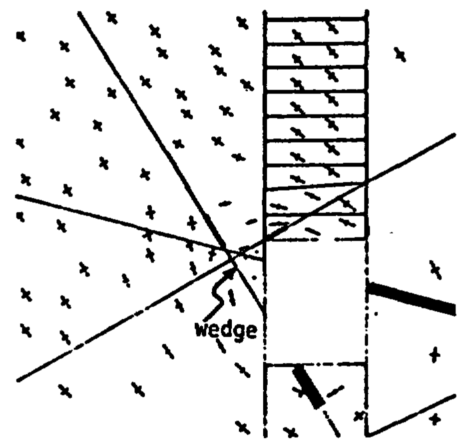
σ_1 = 80 MPa at N55W

$\sigma_1/\sigma_2 = 2.0$

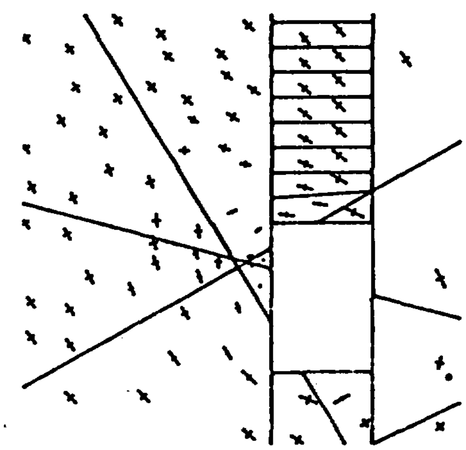
A series of parameter studies were conducted in which the friction angle of the fault and dike were varied from 20° to 10°. The results described here are for a dike friction angle of 10° and a fault friction angle at 17°. In each case, the initial block system is allowed to come to equilibrium under the in-situ stresses, followed by extraction of the 200 panel in a sequence of blasts by removing the panel blocks (Fig. 7) and stepping to equilibrium. Figures 8(a) to (d) show the sequential incremental displacement for the fault and dike structures as the 200 panel is excavated.



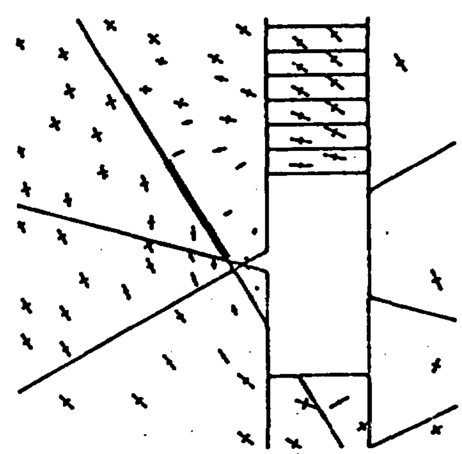
(a) October 11 Blast (Shear occurs on the fault and dike structure in the hangingwall, approximately 4cm maximum displacement.)



(b) October 19 Blast (Shear occurs in the footwall and hangingwall; wall wedge becomes detached.)



(c) November 1 Blast (There is an absence of shear on both structures.)



(d) November 8 Blast (There is renewed slip on dike as normal stress is reduced across its surface.)

Fig. 8 Sequential Plots of Scaled Incremental Shear Displacements on the Fault and Dike Structures As Mining Occurs (The thickness of the line superimposed on the structure indicates the relative shear displacement across the feature.)

The first production blast (October 11) results in significant slip (approximately 4cm) on both structures and is confined to the hangingwall. The second production blast (October 19) results in an equally large incremental slip on the hangingwall fault structure. A large, highly local slip occurs in the footwall of the fault and dike at their intersection. A maximum incremental slip of over 4cm occurs at that location. The footwall block formed by the intersection of the two structures is kinematically freed by this excavation and destressed (Fig. 9).

The incremental shear displacement resulting from the second production blast may be compared to the microseismic activity for the same excavation stage. Figure 6 is the compilation of events occurring in the six hours after the 19 October production blast. These events fall along the projected extent of the off-shoot fault structure. In both the model and field examples, the incremental slip is confined primarily in the hangingwall, although the model indicates slip on both the fault and dike structures. The model indicates the shear displacement along the fault decays to less than 10mm in a radius of approximately 130m from the stope, which is comparable to the radius of influence of the microseismicity. The model shows that a large magnitude slip occurs in the footwall near the fault-dike intersection when the October 19 blast is taken, corresponding to the occurrence of a flurry of seismic activity along both structures following the blast.

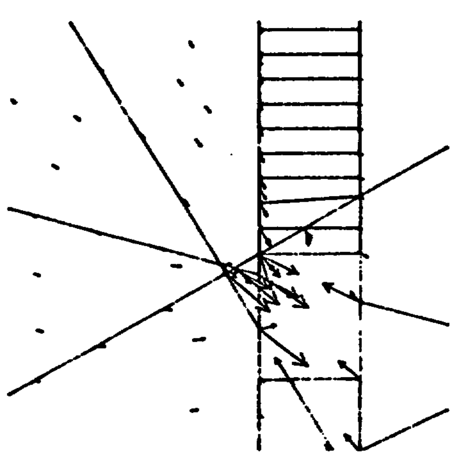


Fig. 9 Wedge Displaces When Kinematically Freed (This figure shows displacement vectors at finite difference nodes.)

The Step 3 excavation (November 1 blast) shows virtually no slip on either structure and correlates with a lack of seismicity monitored at this time. The Step 4 excavation (November 8 blast) results in a large slip centered on the dyke and fault structures in the footwall. This correlates with a large identifiable footwall burst (2.2 M_n) which followed the blast. Damage from this burst was centered along the extension of the fault structure at its intersection with level development. The large wedge (150m³) formed by the fault and dike intersection was expelled into the stope with this event. The slip is related to reduction of normal stress across the structures caused by the excavation.

Conclusions From the 2-D Analysis

The two-dimensional studies show that the major sequence of events (initial conjectured slip on the dike prior to the 200 panel excavation, hangingwall slip on dike and fault prior to and immediately following the 19 October production blast, localized slip and detachment of wall wedges in the 23-200 panel, significant slip following the November 8 blast) are borne out by the numerical model.

The limitations of this analysis are as follows.

1. Only a very crude representation of the truly three-dimensional mining geometry is obtained.
2. It is not possible to determine a three-dimensional picture of the zone of slip from this analysis.
3. The frictional and cohesion properties of the structures must be made artificially low in order to produce slip.

The analysis is worthwhile in that the proper orientation of the in-situ stresses and a lower bound on the fault and dike properties have been determined as initial input to the three-dimensional analysis which is currently being performed.

The model is being used to analyze the completion of excavation of the main sill. Predictions of the slip potential of these structures with further excavation are currently underway, with the ultimate objective of determining the optimum extraction sequence to minimize slip or, at least, to have it occur at the most advantageous time. Other studies with MUDEC are examining the potential for hydraulic stimulation of incremental slip, thereby releasing energy at a slow and pre-determined rate.

CONCLUSIONS

The philosophy of numerical model use at Falconbridge, Ltd. is oriented toward practical solution of mining problems through the use of personal computers. Reliance is not made on one particular model type for analysis of all problems. A number of codes, including two- and three-dimensional boundary element, finite difference, and distinct element codes, are available, depending on problem requirements. In all cases, the numerical modeling must not be performed in a vacuum—it requires detailed comparison to field data and interaction with mine ground control engineers. As this process continues, greater confidence will be placed in the modeling effort.

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