CHARACTERIZATION OF THE HYDROGEOLOGY AND STRESS STATE IN THE VICINITY OF THE HOMESTAKE MINE, LEAD, SD

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CHARACTERIZATION OF THE HYDROGEOLOGY AND STRESS STATE IN THE VICINITY OF THE HOMESTAKE MINE, LEAD, SD

A Thesis
Presented to
the Graduate School of
Clemson University

In Partial Fulfillment
of the Requirements for the Degree
Master of Science
Hydrogeology

By
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Accepted by
Dr. Lawrence C. Murdoch
Dr. Ronald Falta
Dr. James Castle
ABSTRACT

Underground workings in fractured rock are common worldwide. They have applications in numerous areas and fields of study. These include mining operations, civil engineering projects like tunnels and underground facilities, and research projects that require underground laboratories such as the physics research being conducted by Sanford Laboratory at the former Homestake mine and Fermi Laboratory near Chicago (Bahcall et al. 2001, Elsworth 2009, Sadoulet et al. 2006, bge science@DUSEL, fnal.gov). These excavations can reach several kilometers in depth including the 3.9-km-deep TauTona mine in South Africa, the 3-km-deep LaRonde mine in Quebec and the 2.4-km-deep Homestake mine in South Dakota. Large quantities of rock are removed when constructing deep excavations, for example Rahn and Roggenthen (2002) estimated the total volume of rock removed from the Homestake mine to be $2.1 \times 10^7$ m$^3$. Removing large volumes of rock alters the local stress state and ground water flow, potentially increasing risks to workers and the environment (Kaiser et al. 2008, Blodgett et al. 2002, Lucier et al. 2009, Goldbach 2010, Kang et al. 2010).

The objective of this research is to develop a better understanding of how deep rock excavations can alter groundwater flow, stress state, and deformation in the rock that envelopes them. The approach is to evaluate how the hydraulic head, flow paths and stress state have been affected by excavation at the Homestake mine in Lead, South Dakota, one of the deepest mines in North America. The Homestake mine was selected as a focus of this research because it has recently been evaluated as the site of a deep underground research laboratory where an understanding of the groundwater flow and
stress state was needed to plan underground experiments. The investigation includes poroelastic modeling of the Homestake mine using available geologic and geophysical data and mine records.

Results from the analyses indicate that mining and dewatering have changed the hydrology and stress state in the vicinity of the Homestake mine. Dewatering reduces the hydraulic head and changes the flow systems in the vicinity of the mine. Four major hydrogeologic zones are recognized: 1.) a Shallow Flow System in the upper few hundred meters that dominates recharge and discharge to streams, 2.) a Recharge Capture Zone where water that has entered the region as recharge since mining began is captured by the mine, 3.) a Storage Capture Zone where water from storage in the host rock around the mine is captured, and 4.) a Mine Workings Zone where rock has been removed. Water enters the system at the top of the Shallow Flow System and either discharges to the streams or flows downward and becomes recharge to the lower capture zones. The Recharge Capture Zone grows with time as regions of storage are depleted and new recharge enters, and eventually it is assumed that the entire capture zone for the mine will become the Recharge Capture Zone. Fluxes from the Shallow Flow System to the Recharge Capture Zone typically range from $1 \times 10^{-9}$ to $4 \times 10^{-9}$ m/s. The largest recharge fluxes from the Shallow Flow System to the Recharge Capture Zone occur above the shallowest portions of the mine. Recharge flux also occurs above areas adjacent to the mine, and when projected to the surface the Recharge Capture Zone creates a roughly elliptical shape that is 6 km x 3.6 km. The Storage Capture Zone extends out beyond and below the Recharge Capture Zone and when projected to the surface creates a roughly
elliptical region that is approximately 8.3 km x 6.6 km and extends down to depths of almost 5 km. Hydraulic heads and flow paths have been affected beyond the Storage Capture Zone but this water had not reached the mine by 135 years and therefore these regions are not included in the capture zones.

The model was calibrated using in-situ stress data at various points in the mine to improve its ability to estimate the stress state and mechanical deformation around the Homestake mine. This was done by varying the rock density, Poisson’s ratio, the effective Young’s modulus of the workings region, and including initial stresses until predicted stresses best fit in-situ stress data. The changing mechanical properties in the workings and dewatering cause changes to the stress around the mine. The mining process typically causes increased compression laterally around the workings and decreased compression above, below, and within the workings. The greatest changes in total stress are near the base of the mine and reach roughly 40 MPa between the ore bodies and in the lower portions of the West Ore Body.

The softening of the mine region because of material removal and decreased fluid pressure in the workings results in deformation in the vicinity of the mine. Subsidence occurs above the mine region and is greatest near the surface and decreases with depth; above the shallowest workings subsidence can reach approximately 0.18 m. There is also uplift along the footwall of the workings in the deeper portions of the mine that can reach up to 0.022 m. Horizontal displacements of as much as several centimeters occur around the mine and with displacement towards the workings region. Deformation in the
vicinity of the mine results in tilt that is towards the workings with the greatest tilts near the surface.

A fault that intersects the West Ore Body was considered as a location for an experiment into the mechanics of earthquake nucleation, so the stress state in the vicinity of this feature was of particular interest. This simulation shows that mining and dewatering reduce fluid pressure and change stresses along the fault. The shear stress along the fault typically increases along most of the fault and decreases in the region where the fault and West Ore Body intersect. Increased shear is typically on the order of 1 to 2 MPa but can reach as much as 5 MPa in areas around the intersection of the fault and West Ore Body. In the region along the fault intersecting the West Ore Body, the decrease in shear can reach -11 MPa. The total normal stress along the fault becomes more compressive along most of the fault and less compressive in the intersection between the fault and West Ore Body. The increase in total compression is approximately 2 MPa, and the reduction in compression in the intersection is approximately 10 MPa. The critical shear stress along the fault was calculated using Mohr-Coulomb failure criteria presented by Byerlee (1978), and the ratio of the estimated shear stress along the fault and the critical shear stress ($t_s/t_f$) was found to approximate the potential for slip along the fault. Mining results in a reduction in slip potential with values of $t_s/t_f$ ranging from 0.66 to 1.1 before mining and from 0.22 to 0.67 after mining. This reduction in slip potential results from reductions in fluid pressure and increased normal compression caused by mining activities.
DEDICATION

To my parents, sisters, brother, and four best friends.
ACKNOWLEDGMENTS

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INTRODUCTION

Construction and mining operations continue to dig deeper into fractured rock masses around the world. Mines, underground construction projects and labs can be built at depths of several thousand meters with extensive tunneling and excavations for their operations (Murdoch et al. 2012, Kang et al. 2010, Rahn and Roggenthen 2002, SNOLAB.ca, Li et al. 2007, Lucier et al. 2009). These excavations can alter the local hydrogeology, perturb stresses, cause deformation and may induce seismicity (Kaiser et al. 2008, Blodgett et al. 2002, Rahn and Roggenthen 2002, Lucier et al. 2009, Goldbach 2010). These effects pose potential risks, and as projects move deeper there is increasing need to understand the hydrogeology and hydromechanical behavior of deep fractured rock masses. Furthermore, there is need to understand how the hydrogeology is altered by anthropogenic operations and excavations and how it affects deformation and stress state in the local rock. In addition, advancing understanding of the hydrogeology of deep fractured rock masses can provide benefits to other fields, such as CO2 sequestration, deep well drilling, and geothermal reservoirs.

Better understanding the impacts of deep rock excavations will allow for appropriate planning when designing, constructing, and operating these deep rock projects. Safety is of paramount concern with significant injuries attributed to rock failures in underground excavations (WCA 2011, Iannacchione et al. 1999, Mark and Iannacchione 2001). By understanding how deep rock excavations alter the hydrogeology and stress state, better safety planning and preemptive measures in potential high risk areas can be taken. Incoming groundwater is another concern during construction and operations (Atkinson
et al. 1984, Rahn and Roggenthen 2002, Goldbach 2010). Identifying possible changes to flow paths and hydraulic heads that may occur will allow for more comprehensive designing of dewatering systems. Furthermore this knowledge will help identify and plan for alterations to the groundwater flows systems around deep excavations.

Existing deep rock excavations such as the former Homestake mine in Lead, SD, are useful sites when studying the aforementioned effects from large excavations. Extensive data have been collected at the mine during construction, operations, and in preparation for its use as the location for the Deep Underground Science and Engineering Lab (DUSEL). The Homestake mine provides access to large portions of the subsurface because of its extensive network of tunnels, and the geometry of the mine’s workings is well documented. The geology within and around the Homestake mine has been studied in considerable detail (Caddey et al. 1990, Redden and French 1989, Dodge 1942, Noble and Harder 1948, and Golder Assoc. 2010). Studies by Rahn and Roggenthen (2002), Davis et al. (2003), and Murdoch et al. (2012) have been done to better understand the hydrology around the Homestake mine and how mining has affected the hydrology. Work has also been carried out by Pariseau (1985), Johnson et al. (1993), Girard et al. (1997), and Golder Assoc. (2010) to estimate in-situ stresses as well as in-situ mechanical parameters, such as the Young’s and Shear moduli and Poisson’s ratio around the Homestake mine. The wealth of knowledge and data on the Homestake mine make it a valuable asset when studying the hydrogeology and hydromechanics of deep rock excavations and a pristine location for research endeavors such as DUSEL.
Objective

The primary objective of this research is to understand how the hydrogeology and stress-strain state have been affected by the creation of the Homestake mine.

Approach

The approach to complete the objectives includes modeling of the Homestake mine using available data. The Homestake mine was chosen for the model because of its depth, approximately 2.4 km, and because of the data available from the site including historical records, previous research studies, and geophysical data collected during planning for the DUSEL project. Modeling involves developing a simulation that couples the hydraulic head and stress state in the vicinity of the Homestake mine. The geometry of the mine workings and hydraulic boundary conditions were taken from literature and records from mining operations. Preliminary estimates of hydraulic and mechanical properties of the rock was based on available information, and then adjusted by calibrating the model to predict the observed dewatering rates and in-situ stresses. Using the calibrated parameters the analysis was simulated without the effects of the mine under steady state conditions to determine the pre-mining hydrology and stress-strain state. The analysis was then run under transient conditions to determine the effects to the hydrogeology and stress-strain state due to mining. The pre-mining and post-mining models were compared to estimate changes to the hydrogeology, stress, and strain that have occurred because of excavations, re-filling of the mine after operations ended, and dewatering of the mine in preparations for DUSEL. Furthermore, through
calibration, the model provides estimates of the hydromechanical properties of the mine region including hydraulic conductivity, specific storage, porosity, fracture compliance, Poisson’s ratio, rock density, and Young’s modulus.

Motivation

1. Improving understanding of the hydrogeology and stress state in the vicinity of the Homestake mine:

The Homestake mine shut down in 2002, and management of the underground workings was transferred in 2006 to the South Dakota Science and Technology Authority to facilitate transition to a scientific research facility. The site was selected in 2007 as the preferred location of the Deep Underground Science and Engineering Lab (DUSEL), an ambitious proposed project with the goal of hosting experiments in physics, biology, geomechanics, and hydrogeology (Bahcall et al. 2001, Elsworth 2009, Sadoulet et al. 2004, bge science@DUSEL). The National Science Foundation decided not to pursue DUSEL in 2010, and soon thereafter the Department of Energy initiated support for physics experiments and an evaluation of long-term applications for Homestake. Today, the facility hosts two major physics experiments, as well as a variety of other research and educational activities.

This study was motivated initially to advance understanding of the hydrogeology and stress-state in the vicinity of Homestake to support the planning of experiments in hydrology, biology and rock mechanics for DUSEL. Planning for DUSEL science was scaled down in 2011 following the decision at NSF to withdraw as a major supporter of the facility. Homestake continues as an important scientific facility, and advancing the
understanding of the groundwater flow and stress state in the vicinity remains a significant motivator for this research.

2. Improving the general knowledge of the hydrogeology and stress state around excavations in deep rock masses:

The geologic setting of the Homestake mine is similar to cratonic regions throughout the world. Understanding the effects of this deep mine in a cratonic setting will provide insights that could improve planning, construction, and maintenance of excavations in similar settings.

**Thesis Organization**

This thesis is organized into four sections. The first introduces the Homestake mine and the hydrology and geology in the vicinity of the mine. The second introduces and describes the development of the 3-D poroelastic model of the mine. The third presents the results from the model. And the final section includes a discussion of the results and conclusion inferred from the numerical analysis.
HYDROMECHANICAL ANALYSIS OF THE
HOMESTAKE MINE, LEAD SOUTH DAKOTA

The Homestake mine is one of the largest and deepest gold mines in North America and the largest gold producer from a single deposit outside of South Africa (Mitchell 2009, Caddey et al. 1990). The mine is located in the Whitewood mining district in Lead, South Dakota, Lawrence County, in the northern Black Hills at lat 44°22' N., long 103°45' W (Fig. 1-1), (Mitchell 2009, Caddey et al. 1990).

Along with being one of the largest gold producers in North America, the mine has a long history as a site of physics experiments. In the 1960s, an experiment led by Ray Davis was developed on the 4850 Level (the Level is so named because it is 4850 ft below a benchmark at the ground surface) to measure the neutrino flux from the sun. The nearly 1-mile-thickness of rock shielded the experiment from the interfering effects of cosmic radiation. The Davis experiment provided important fundamental data on solar neutrino flux, winning Davis a Nobel Prize in physics in 2002 (sanfordlab.org).

The legacy of the Davis experiment was well known to the current generation of particle physicists when it was proposed to consider the Homestake mine as the site for the Deep Underground Science and Engineering Laboratory (Sadoulet et al. 2004, Elsworth 2009, Bahcall et al. 2001). The mine also provides an excellent location to study other scientific phenomenon, including the evolution of life in extreme environments, the interaction between fluid flow and the state of stress, the deformation of large underground rock masses, and the thermal, hydraulic, mechanical, chemical, and
biological interactions and coupling that control the flow of fluids, energy, and nutrients in fractured rock (Bahcall et al. 2001, Elsworth 2009, Sadoulet et al. 2004, bge science@DUSEL).

The Homestake mine was recently chosen as the location for the new Deep Underground Science and Engineering Lab (DUSEL) which will facilitate studies in numerous fields including physics, biology, geosciences, and engineering (Bahcall et al. 2001, Elsworth 2009, Sadoulet et al. 2004, bge science@DUSEL). The former Homestake mine was chosen for the location of DUSEL because of its depth (~2.4 km),

Figure 1-1. Location of the modeled region in the Black Hills with major geographic features. Shaded blue region represents mine workings projected to the surface. Modified from a Google earth image of the region. (accessed 1/20/2013)
extensive workings, and it is accessible for scientific study for an extended period of time (~30 yrs) (Bahcall et al. 2001, Elsworth 2009, Sadoulet et al. 2004, bge science@DUSEL). The large extent of the mine workings allows for large regions of the sub-surface to be accessed for study in several different environments and geologic settings. The depth of the mine workings provides shielding from cosmic radiation for physics experiments, a location to study the geomechanics and hydrogeology of fractured rocks at depth, and a location to study biology and the development of life in extreme conditions (Elsworth 2009, Sadoulet et al. 2004, bge science@DUSEL). Furthermore because the mine can be used for an extended period of time, a better understanding of the temporal variations associated with fluid-rock interactions, reactions of aqueous and gaseous species in the sub-surface and microbial alterations to these species and the surrounding host rock mineralogy at in-situ conditions can be developed (Elsworth 2009, Sadoulet et al. 2004, bge science@DUSEL).

A better understanding of the hydrogeology and stress state with improved insight into the hydromechanical behavior around the mine region will improve the design and planning of experimental studies at DUSEL and is the primary motivation of this thesis. Furthermore, the scientific phenomena mentioned above have important implications for the overall design, construction, maintenance, and safety of underground excavations in deep fractured rock masses. Understanding the interaction between fluid flow and stress state will provide a better understanding of the mechanical effects that occur when large excavations are dewatered. This could improve planning to reduce and mitigate these effects. Furthermore, the thermal, hydraulic, mechanical, chemical, and biological
interactions and couplings play a large part in the movement and deposition of valuable natural resources underground. Better understanding of these interactions and the environment these interactions occur in improves our ability to predict where these resources can be found underground and our ability to remove these resources as efficiently as possible.

**The Mine**

Placer gold was discovered in the southern Black Hills in 1874 by the Custer expedition near the present village of Custer, and then a year later in Deadwood Creek roughly 5 km east of present day Lead in the northern Black Hills (Mitchell 2009, Caddey et al. 1990). In 1876, the placer gold in Deadwood Creek was traced to surface expressions of the Main Ledge and Caledonia ore zones of the Homestake deposit (Mitchell 2009, Caddey et al. 1990). After this discovery, an open-cut surface mine was started, and early mining activities followed the Main Ledge and Caledonia ore ledge down-plunge, using shrinkage and timber stoping methods without backfilling (Mitchell 2009, Caddey et al. 1990). This method was replaced by drawhole mining, an uncontrolled block-cave system, which eventually resulted in mined areas caving to the surface (Mitchell 2009, Caddey et al. 1990). During modern times, the Homestake mine used mechanized cut-and-fill and vertical crater retreat mining methods, which are safer than earlier methods (Caddey et al. 1990).

There are nine productive ore deposits, or *ledges*, in the mine. The history of their discovery, gold production and significant events are summarized in (Fig. 1-2). Mining continued for 126 years since the discovery of gold in 1876 until mining stopped in 2002.
Figure 1-2. Ore ledge discovery, annual gold production, and significant historical events, Homestake Mine, Lead, S.D. From Caddey et al. (1990).
The mine consists of roughly 500 kilometers of tunnels and stopes contained within a region that encompasses approximately 5 cubic kilometers (Rahn and Roggenthen 2002). Mine depth increased at a roughly uniform rate of 26 m/yr until the mine reached a total depth of 2,440 m (8,000 ft) below ground surface by 1975 (Fig. 1-3), (Rahn and Roggenthen 2002). The mine developed as a complicated network of vertical stopes, shafts, and boreholes and horizontal tunnels and workings advancing with depth and laterally as new ore was discovered and excavated. The mine region in cross-section resembles an irregular column that plunges approximately 40° to the SE (Fig. 1-5). The individual horizontal levels in the mine are
separated by regions of rock 30 m thick at shallow depths and 45 m thick below depths of 300 m (Murdoch et al. 2012). The region of rock separating each level is cut through with numerous vertical stopes, boreholes, and shafts (Murdoch et al. 2012). The two main shafts used to access the mine are the Yates and the Ross shafts. The Yates shaft is located to the east-southeast of the open pit region at a coordinate of roughly (1220 m, 760 m), with respect to the Homestake coordinate system, or a latitude/longitude of 44°21’6.1’’/-103°44’58.7’’. The Ross shaft is located approximately 850 m from the Yates shaft towards the SW. The Homestake coordinate system is used by the mine and researchers at the mine to reference location and has an origin (0,0) at a latitude/longitude of roughly 44°21’32.06’’/-103°45’58.06’’. This coordinate system is also used to reference locations within this numerical simulation.
During mining, pumps were used to dewater the mine to prevent flooding, and the water level is assumed to roughly follow the bottom of the mine with time (Fig. 1-3). After mining ended, the pumps were shut off in 2003 and the water level in the mine rose over the next five years and was tracked by monitoring the times when the electrical circuits shorted out at various levels (Murdoch et al. 2012). Dewatering was commenced again in 2008 when the mine was chosen for the site of the DUSEL project. The fall in water level was measured with a pressure transducer installed at depth (Davis et al. 2009, Zhan and Duex 2010). The underground workings have been managed since 2006 by the South Dakota Science and Technology Authority, which oversees the Sanford Underground Research Facility (sanfordlab.org). Several physics experiments are currently on-going, and a variety of others are being planned or proposed.
**Geology and Setting**

The geology of the Black Hills is characterized by a major structural dome surrounded by smaller domes that step into rolling hills and the plains and basins of the northern Great Plains. The elevations in the Black Hills range from approximately 915 m (3,000 ft) above sea level in the plains flanking the central uplift to a maximum of 2,207 m (7,242 ft) at Harney Peak (Driscoll et al. 2002). Ground surface elevations near the former Homestake mine range from approximately 1500 to 1700 meters above sea level.

**Regional Geology**

The Black Hills uplift in western South Dakota is an elongated dome approximately 85 km long and 35 km wide that consists of a Precambrian core flanked by Paleozoic and younger sediments (Fig. 1-6), (Darton et al. 1925). The rocks of the Black Hills uplift range in age from Archean to Holocene. A minor portion of the core consists of Archean granitic rocks at least 2.5 Ga, which occur in the Little Elk Creek (Zartman and Stern 1967, Gosselin et al. 1988) and Bear Mountain areas (Ratté and Zartman 1970).
The majority of the Precambrian core consists of early Proterozoic sedimentary and volcanic rocks regionally metamorphosed from greenschist to upper amphibolite facies that range in age from 2.20 to 1.87 Ga (Caddey et al., 1990, Redden et al. 1990). Protoliths of the Early Proterozoic metasedimentary rocks consist of clastic sediments,
black shale, iron- formations, and volcanic rocks (Caddey et al. 1990). The Precambrian core of the Black Hills is unconformably overlain by Paleozoic and Mesozoic sedimentary deposits as well as Tertiary surficial deposits. The Precambrian, Paleozoic, and Mesozoic rocks of the northern Black Hills are intruded by many early Tertiary alkali-calcic to alkali stocks, laccoliths, dikes and sills (Caddey et al. 1990).

Deformation of the rock in the Black Hills region is complex, with at least four events resulting in extensive folding and faulting (Caddey et al. 1990). The most intense of these events was the first, which resulted in the large-scale structural pattern of the region (Fig. 1-7), (Caddey et al. 1990). Regional deformation and metamorphism of the Archean and Early Proterozoic rocks occurred prior to the emplacement of Harney Peak Granite (1.72 Ga), the youngest Proterozoic rock in the Black Hills (Redden et al. 1990). Studies at the Homestake mine suggest that the regional metamorphism was syndeformational with regional foliation in the Lead area (Caddey et al. 1990). By 530 Ma, the Precambrian core of the Black Hills had been uplifted and eroded, and on this surface Paleozoic and Mesozoic sediments were deposited during a period of relative tectonic quiescence (Caddey et al. 1990). The present-day domal configuration of the Black Hills was created during the Laramide Orogeny (65-60 Ma), which resulted in regional uplift and subsequent erosion of sediment-covered Precambrian basement blocks (Caddey et al. 1990, Tweto 1975). Tertiary intrusions into the Precambrian, Paleozoic, and Mesozoic rocks occurred after the Laramide uplift (60-50 Ma). Renewed erosion resulted in re-exposure of the Precambrian core, and during the Oligocene a thin layer of terrestrial sediments was deposited over much of the Black Hills (Caddey et al. 1990).
Subsequent erosion and surficial deposition led to the current topography of the Black Hills region (Caddey et al. 1990).

The foliation in the vicinity of the Homestake mine strikes roughly N20W and dips steeply to the NE (Rahn and Roggenthen 2002). Ductile shear zones in the region are oriented roughly parallel to the regional foliation (Caddey et al. 1990).
**Local Geology**

The Homestake Mine is located at the northern end of the regional uplift of the Black Hills, within the eroded core of an early Tertiary dome known as the Lead window. The dome appears to have been formed by emplacement of the Cutting stock and many dikes and sills of alkali-calcic composition, all of Tertiary age. The Cutting Stock is 1.2 km wide by 2.5 km long. Rhyolite dikes of Tertiary age are exposed in the Homestake mine, and may be related to emplacement of the nearby stock (Caddey et al. 1990, Redden and French 1989). Earlier studies by Dodge (1942) and Noble and Harder (1948) divided the Early Proterozoic rocks of the Homestake region into six formations (Fig. 1-9); though only three formations are significant to the Homestake mine. These include the Poorman Formation, the Homestake Formation, and the Ellison Formation.

The Poorman Formation is the oldest and is comprised of a lower unit of metatholeiite, known as the Yates unit, overlain by an upper unit of heterogeneous, calcareous, pelitic to semi-pelitic phyllite (Caddey et al. 1990). The Homestake Formation is a carbonate-facies iron-formation (Caddey et al. 1990). The youngest of the three formations, the Ellison Formation, consists of quartzite and pelitic to semi-pelitic phyllites. All three units are isoclinally folded and metamorphosed (Nobel et al. 1949, Caddey et al. 1990). These formations are shown in plan-view at the surface in (Fig. 1-7), and in cross-section in (Fig. 1-10).
The region around the Homestake mine is highly deformed with intense folding that ranges in scale from several kilometers to centimeters (Murdoch et al. 2012). The general structure around the Homestake mine includes the Lead anticlinorium and synclinorium and the Poorman anticlinorium to the west. The axes of these structures plunge roughly 35°-50° to the southeast. The rocks are further deformed into tight, isoclinal folds on scales from centimeters to hundreds of meters (Fig. 1-8), (Caddey et al. 1990). A fault was discovered during recent work at the Homestake mine that was exposed at several levels within the mine (Germanovich et al. 2011). It was estimated that the fault had a strike of roughly N20W and dip of 60° NE (Germanovich et al. 2011). The fault is approximated to be roughly 1.5 km along strike and dip with a center that was roughly 1.5 km below ground surface (Germanovich et al. 2011).
Figure 1-9. Schematic of the stratigraphy of the Homestake region. From Caddey et al. (1991).
Figure 1-10. General geologic cross-section A-A’ through 33 Stope, Main Ledge Reference Line, Homestake mine. Cross-section cut shown in (Fig. 1-9). Modified from Caddey et al. (1991).
Climate of the Black Hills

The climate of the Black Hills region is continental with severe seasons of warm summers, cold winters, and heavy fluctuations in precipitation and temperature (Johnson 1933). The long-term precipitation average in the Black Hills is 0.47 m/yr and ranges from 0.26 m/yr in 1936 to 0.7 m/yr in 1955 (Fig. 1-11), (Driscoll et al. 2000).

Figure 1-11. Precipitation and variation from the long-term mean for the Black Hills, from 1931-1998. From Driscoll et al. (2000).
Precipitation usually increases with altitude and is greatest in the northern Black Hills near Lead. Moist air currents from the northwest affect the northern Black Hills, whereas drier currents from the south-southeast affect the southern Black Hills (Fig. 1-12). The greatest precipitation usually occurs from May to June and the least from November to February (Driscoll et al. 2002). The average annual temperature is 6.6°C and ranges from approximately 2.8°C around Deerfield Reservoir to 9.3°C near Hot Springs, average temperature typically decreases with increasing altitude (Driscoll et al. 2000).
Hydrology

The region above the Homestake mine supports perennial streams that are spaced on the order of one kilometer (Carter et al. 2002, Murdoch et al. 2012). Streams typically have headwaters near the base of the limestone plateau, and then they flow over the exposed Precambrian core of the Black Hills (Fig. 1-13) (Driscoll et al. 2002). Streams in the Black Hills lose much of their flow after passing over outcroppings of the major limestone aquifer units that ring the Precambrian core (Fig. 1-13). These carbonate outcroppings are weathered with high porosities and permeability, and infiltration rates are exceptionally high (Driscoll and Carter 2004, Driscoll et al. 2002). Water that is not lost as recharge to aquifers typically flows to larger river systems along the flanks of the Black Hills, where it then flows out of the region to the east. Perennial streams in the

Figure 1-13. Schematic of water flow in the Black Hills. Solid black arrows show general flow directions. From Driscoll et al. (2002).
vicinity of the Homestake mine include Whitewood Creek and its tributaries including Whitetail Creek, Deadwood Creek, City Creek, Gold Run Creek, West Strawberry Creek, and Yellow Creek, as well as Spearfish Creek, False Bottom Creek, Polo Creek, Miller Creek, Elk Creek, Bear Butte Creek, and Two-Bit Creek. There is a pond near the mine where the tailings wash and water from dewatering the mine were discharged. This pond is located south-east of the open pit almost directly above the lower reaches of the Main and East ore bodies.

Once water enters the sedimentary rocks ringing the exposed Precambrian core, it can follow many possible paths. The majority of water within the system is believed to discharge under the basins and plains east of the Black Hills region. Some water leaks through upper confining layers and returns to the surface as artesian spring flow at lower altitudes (Driscoll and Carter 2004, Driscoll et al. 2002).

**Hydrologic Budget**

Hydrologic conditions around the former Homestake mine include an active surface hydrology overlying low permeability formations with minimal ambient flow (Driscoll and Carter 2001, Davis et al. 2003, Rahn and Roggenthen 2002, Murdoch et al. 2012). The average precipitation for the region, which is the wettest in the Black Hills, is approximately 0.7 m/yr (2.22x10^-8 m/s) (Driscoll and Carter 2001, Driscoll et al. 2002). The surface hydrologic system supports perennial streams spaced at approximately 1 km, which are fed by both groundwater discharge and stormflow. Baseflow to these streams ranges from 40-70% of their total flow. A hydrologic budget was done on a 10-year record of stream flow from Whitetail Creek (station 06436156) (Fig. 1-14) (Driscoll and
Carter 2001). The stream is located a few km southwest of Lead and drains a 15-km² region with an average total flow (normalized to watershed area) of $8.5 \times 10^{-9}$ m/s. This flow is roughly one third (0.37) of the average precipitation, and base flow is $5.2 \times 10^{-9}$ m/s (Driscoll and Carter 2001). A hydrograph baseflow separation technique was performed by Murdoch et al. (2012) to confirm these values.

The long-term average recharge flux is assumed to equal the average baseflow of $5.2 \times 10^{-9}$ m/s. This assumption implies that groundwater does not cross over watershed boundaries (Murdoch et al. 2012). The recharge rate can vary from the baseflow in regions where the groundwater divide is different than the surface water divide, which occurs in areas of the Black Hills underlain by karstified limestone. The majority of the Whitetail Creek watershed is underlain by metamorphic rocks, however, so this effect seems to be unimportant in the vicinity of the Homestake mine (Driscoll and Carter 2001, Murdoch et al. 2012).

**Hydrogeologic Conceptual Model**

The hydrogeologic conceptual model consists of a dome of low permeability rocks partially overlain and flanked by more permeable rocks. The surface hydrology consists of active perennial streams that are spaced on the order of km (Fig. 1-15) (Carter...
et al. 2002). The surficial material is assumed to be more permeable than the underlying rocks. This surficial material could be sedimentary/carbonate rocks overlying the metamorphic basement rock or metamorphic rocks that have higher fracture densities or fracture aperture openings than the metamorphic rock at depth. The recharge rate to the shallow flow system is assumed to be consistent with baseflow observed in Whitetail Creek.

Figure 1-15. Hydrogeologic conceptual model of the vicinity of the Homestake mine. Shows the shallow hydrologic system overlying the deep system controlled by dewatering. Thick solid arrows are flow lines originating as recharge, thin solid lines are flow released from storage, dashed lines are flow along the fault plane.

Individual tunnels and stopes in the mine workings are represented using effective properties following the approach outlined in Murdoch et al. (2012). This approach considers ground water flow through two overlapping continua, one representing the workings and another representing the rock between the workings. The
tunnels and stopes are assumed to reduce the effective elastic modulus of the vicinity of the workings.

The conceptual model recognizes the geometry of 5 mined-out regions and one large fault (Fig. 1-15). This is a significant refinement from the model described by Murdoch et al. (2012), where the workings were represented as a single region. This refined geometry is needed to resolve the vicinity of the regions where in-situ stresses were measured and to characterize the hydromechanical role of the fault.

It is expected that reducing the water level in the mine will result in groundwater inflow into the mine region. The groundwater capture zone will then expand outward to affect the general vicinity of the mine as the water level continues to fall. Continued mining and dewatering would induce downward flow of water from the shallow flow system toward the mine as well as flow from greater depths where water is being removed from storage (Fig. 1-15).
NUMERICAL MODEL

A numerical analysis was developed to gain insight into the hydrogeology and stress state around the Homestake mine and to provide additional information on the hydrogeologic conceptual model (Fig. 1-15). This analysis builds on a previous study performed by Murdoch et al. (2012) and uses the same governing physics with the inclusion of fully anisotropic conditions for the mechanical portion of the analysis, improvements to the geometric features, and improved calibration.

The numerical analysis was performed using COMSOL Multiphysics 3.5a. COMSOL is a multiphysics modeling package distributed by COMSOL, Inc. that is capable of solving coupled physics analyses simultaneously utilizing an underling system of partial differential equations (PDEs) (COMSOL Multiphysics User’s/Modeling Guide). The system of PDEs is solved using the finite element method (FEM) which approximates the PDE analysis with a system of equations that have a finite number of unknown parameters (COMSOL Multiphysics User’s/Modeling Guide). This is done by first discretizing the geometry of the original problem, also known as meshing, which partitions the original geometry of the analysis into small units of simple shapes. Meshing results in a finite number of shape elements that closely approximate the original geometry (COMSOL Multiphysics User’s/Modeling Guide). Once the model has been discretized approximations of the dependent variables within each shape element are compiled. Using this method the dependent variables are approximated using functions that can be described with a finite number of parameters or degrees of freedom (DOF) (COMSOL Multiphysics User’s/Modeling Guide). This numerical
analysis uses finite elements that are 2nd order Lagrange elements and therefore the dependent variables are estimated using quadratic equations defined for each DOF in a given discretized shape element. Inserting these approximations into the weak form of the PDEs generates a system of equations that approximate the dependent variables in the discretized shape elements of the model (COMSOL Multiphysics User’s/Modeling Guide). The systems of equations for the shape elements and the imposed boundary conditions and constraints are then used to iteratively solve for the dependent variables within the numerical simulation. The method for solving PDEs used in this study is often known as the Galerkin finite element method. This study used two coupled PDEs when modeling the Homestake mine, a stress-strain PDE to simulate mechanical stress and deformation and a heat equation PDE to simulate fluid flow and pressure. A heat equation PDE could be used to simulate fluid flow and pressure because from a mathematically point of view the mechanics of heat flux are the same as fluid flux.

The model represents the region around the mine as a 3-D domain that is roughly 25 km by 25 km by 10 km deep with a top elevation of 1570 m above sea level. The lateral extent of this region stretches from Elkhorn peak in the north to Custer peak in the south and from approximately 1 km east of the westernmost portion of highway 14A to approximately 7 km west of Sturgis, SD (Fig. 1-1). The mined-out region is represented as 3-D domains corresponding to the major ore bodies. Each of these areas is characterized during the flow analysis as two superimposed domains of different properties; one domain represents the mined-out material and the other represents the un-mined rock. Water is allowed to exchange between the domains. The analysis uses a
coupled poroelastic approach. The analysis simultaneously computes the change in fluid pressure and the deformation resulting from this pressure change as well as the deformation from changing stresses and the change in fluid pressure this causes.

**Model Geometry and Mesh**

The model geometry includes a base region with an overlying surface region. Major ore bodies, surface water features including streams and the tailings pond, a fault that cuts through the lower workings of the mine, and the major shafts of the mine are represented as subdomains. The base region is a 25x25x10 kilometer block with a surface region that encompasses the upper 250 meters of the model area (Fig. 1-16).

![Figure 1-16](image.png)  
**Figure 1-16.** Base region of the poro-elastic model, showing base block, surface region, surface waters, open pit region, and cylindrical area for mesh control. Cross-section A-A’ shown in (Fig. 1-17).

The model subdomains are referenced to the coordinate system used to describe the geometry of the mine workings in order to facilitate interchange between field data and model parameters and results. The base region is horizontally centered about the origin of the mine coordinates (0,0). The top elevation of the base region is at 1,570 meters. The top elevation was chosen to approximate the average surface elevation
above the mine region as calculated from USGS DEM data of the modeled region and adjusted so the hydraulic head at the top surface above the mine did not exceed the elevation of the top surface above the mine.

The coordinate system and origin (0,0,0) of the numerical model corresponded to those used by Sanford Laboratory and is the coordinate system used to designate in-situ stress measurement locations in the mine. This was done so that during calibration the predicted stresses from the model and the measured in-situ stresses at given points in the mine region could easily be compared.

Figure 1-17. Cross-section A-A’ from (Fig. 1-16), showing ore bodies, open pit, fault plane, and major shafts.
Ore Bodies

The regions representing the ore bodies were delineated using cross-sectional borehole data (Fig. 1-18) and plan view maps of the workings at the major levels in the mine (Appendix A: from the Vulcan Database). The cross-sectional borehole data contained (x,y,z) coordinates (Fig. 1-18) that delineated the vertical extent of the major ore regions and direction of their major axes. This gave both the shape of the ore regions and the orientation of their axes in reference to the Homestake coordinate system. The coordinates were imported into the visual CAD program, AutoCad, to create cross-sections that could be extruded to create the ore regions. The width of the extrusion for each ore region was estimated from the plan view maps. This was done by estimating the...
average width of the dense region of workings at each level. As a result, the ore-bearing zones are defined based on the mapped tunnels and stopes, rather than on the ore-grade. The width of the workings of each ore body was measured at each level, and then the average width for each individual ore body was calculated. The cross-sections of each ore body were then extruded using the widths found with the method described above to create 3-D geometric shapes representing the major ore regions. This method gave an average width of 150 meters for the Main, West, and 13&15 Ledge ore bodies and an average width of 450 meters for the East ore body. Chamfers were applied to the sharpest corners of the geometries for the two largest ore bodies to improve the mesh quality and reduce the overall number of mesh elements generated for these regions.

**Streams and Tailings Pond**

The streams and tailings pond were delineated using a combination of USGS DEM data and USGS WaterWatch mapper and Google Earth imagery. The DEM data was plotted in 3-D to determine the primary channels. Then images from WaterWatch mapper and Google Earth were used to determine which of these channels were rivers and creeks.

Figure 1-19. The streams and tailing ponds used in the numerical analysis with major stream labels.
Using this method the major coordinates that delineate the surface waters were found and then converted into the Homestake coordinate system (Fig. 1-19). To limit the mesh density at the surface of the model the resolution of the streams was reduced away from the mine region by removing the small-scale meanders. The coordinates defining the tailings pond were found using the same method giving a rough outline of the pond.

**The Homestake Fault and the Shafts**

The location and extent of the Homestake fault in the model was estimated by correlating contacts of the fault at various levels in the mine. This was done using geologic maps of the major levels in the mine which showed the fault contact at various levels. Using this method the orientation, location and size of the fault surface could be estimated. The fault was assumed to be a circular plane cutting through the host rock and ore regions. The diameter of the fault is approximately 1500 meters, with a strike of N20W and a dip of 60° NE. The center of the fault is at (x=152, y=-2134, z=55) [m] in the Homestake coordinate system.

The (x,y) coordinate locations of the major shafts were found on the plan view maps used to delineate the width of the ore regions, and the depths of these shafts were determined from cross-sections of the mine workings.

**Meshing**

The model geometry was meshed using a variable tetrahedral mesh that was generated in several stages to improve meshing quality and resolution around the mine region. First the ore regions and the fault plane were meshed with a maximum element size of 250m.
Then the surface of the model was meshed with a maximum element size of 500m. Meshing around the mine region was controlled by adding an elliptical cylinder in which the mesh size could be controlled (Fig. 1-20). The maximum element size in this cylinder was set to 3000 m. Around the ore regions the mesh in the cylinder was controlled by the mesh element sizes along boundaries of the ore bodies. Outside of the cylindrical subdomain, the maximum element size was not constrained. This meshing scheme created a mesh with elements of roughly 100m scale in the vicinity of the mine, and the maximum element size increased to roughly 5 km in the subsurface at the edge of the model. The size of the elements at the ground surface were smaller than in the subsurface, with element sizes of approximately 200m.
above the ore body region and maximum sizes of approximately 500 m at the boundaries of the model. The mesh had 33,551 elements.

**Governing Physics**

The governing physics of the numerical analysis were based on a previous study of the Homestake mine by Murdoch et al. (2012) and modified to incorporate anisotropic mechanical parameters. The physics of the analysis assume that the workings of the mine and the rock domain are continuous domains that occupy the same space similar to the method presented by Warren and Root (1963). The region that represents the mine workings moves downward with time to represent the effects of mining.

**Rock Domain**

The numerical model simulates both the changes in pressure and fluid flow as well as the mechanical effects around the mine including the change in stress and strain that occur because of mining. This is done using a poro-elastic analysis that combines a hydrologic analysis and a linear elastic stress-strain analysis. The hydrologic analysis used in the model is common in groundwater hydrogeology given by Eq. (1.1) (Bear 1979, Wang and Anderson 1982, Anderson and Woessner 1992).

\[ \nabla \cdot \left( S_s \frac{\partial h}{\partial t} - R \right) \]

where \( h = \frac{P}{\gamma} + z \) is the hydraulic head [basic units: L], \( K \) is the hydraulic conductivity tensor [L/T], \( S_s \) is the specific storage coefficient [1/L], \( R \) is the fluid source term [1/T], \( P \) is pore pressure [M/LT²], \( \gamma \) is the unit weight of water [M/L²T²], and \( z \) is the vertical coordinate [L].
The hydrologic analysis is then coupled to Eq. (1.2) to simulate the elastic deformation that occurs due to pressure change (Biot 1941, Detournay and Cheng 1993, Bai and Elsworth 2000, Wang 2000).

\[
\frac{E}{2(1+\nu)} \nabla^2 u_i + \frac{E}{2(1+\nu)(1-2\nu)} \frac{\partial \varepsilon}{\partial x_i} = \alpha \gamma \frac{\partial P}{\partial x_i}
\]

(1.2)

where \( u_i \) is displacement of the solid in the ith direction [L], \( \varepsilon \) is volumetric strain, \( E \) is the drained Young’s modulus [M/LT^2], \( \alpha \) is the Biot-Willis coefficient, and \( \nu \) is the drained Poisson’s ratio. In Eq. (1.2) \( x_i \) is the spatial coordinates and the ‘i’ subscript is incremented from 1 through 3 to indicate coordinate directions. The material properties in Eq. (1.2) assume the effects of the fractures and solid matrix are lumped together.

**Mine Workings**

The depth of water in the mine, \( h_w \) [L], was taken from Murdoch et al. (2012) (Fig. 1-5). It is assumed that the pressure head within the air-filled workings is zero and the hydraulic head, \( h_m \) [L], is given by Eq. (1.3).

\[
h_m = z (0 > z > h_w)
\]

(1.3)

This is only satisfied if the walls of the workings are covered in a film of water; the head in the workings would be lowered, however, by drying from the ventilation system (Murdoch et al. 2012). Since the head change in the mine is large compared to this reduction in head this effect can be neglected (Murdoch et al. 2012).

The pumps were shut off in 2002 when mining ended, and the mine began filling with water. By 2008 the water in the mine had reached a depth of approximately 1 km
(Fig. 1-5), and it was assumed that the hydraulic head in the flooded part of the mine is equal to the elevation of the free water surface.

\[ h_m = h_w \left( h_{\text{wmin}} < z \leq h_w, t > 2003 \right) \] (1.4)

where \( h_{\text{wmin}} \) is the minimum (deepest) water level (-2,400 m). This approach assumes that the horizontal hydraulic head gradients in the large conduits are negligible and horizontal gradients within the mine workings are omitted from this formulation (Murdoch et al. 2012). This is a reasonable assumption due to high permeability of the mine workings in comparison to the surrounding rock.

### Coupling between Domains

Coupling between the two domains of the model, the enveloping rock and the mine workings, was accomplished using the source term in Eq. (1.1) (Murdoch et al. 2012). The approach resembles that used for dual porosity media with two scales of permeability presented by Barenblatt et al. (1960), Warren and Root (1963), and Elsworth and Bai (1992).

\[ R = C_g \left( h_m - h \right) \frac{K_{\text{avg}}}{L^2} \] (1.5)

The source term, \( R \), was modified to include poroelastic coupling with Eq. (1.6)

\[ R = C_g \left( h_m - h \right) \frac{K_{\text{avg}}}{L^2} - \alpha \frac{\partial \varepsilon}{\partial t} \] (1.6)

where \( L \) is a characteristic spacing between tunnels, \( C_g \) is a constant that depends on geometry, and \( K_{\text{avg}} \) is the average hydraulic conductivity (scalar). The source term, \( R \), controls flow from the enveloping rock to the mine workings and is controlled by the
difference between the head in the mine, \( h_m \), which is known from historical data and the head in the enveloping rock, \( h \), which is calculated in the analysis. Poroelastic coupling results in an additional fluid source that is caused by volumetric strain of the solid and is controlled by \( \alpha \) and the strain rate (\( \delta e / \delta t \)). The workings occupy a region of approximately \( V_w = 1.1 \text{ km}^3 \), and they consist of approximately \( L_d = 500 \text{ km} \) of tunnels, therefore the average spacing is assumed to be

\[
L \sim \frac{V_w}{L_d} = 50m
\]

(1.7)

The shape and spacing of the tunnels control the geometric constant, \( C_g \). The characteristic time required for head affected by adjacent tunnels to interact if the tunnels are parallel and evenly spaced is given by Eq. (1.8) (Murdoch et al. 2012).

\[
\tau \approx \frac{S L_d^2}{K_{avg}}
\]

(1.8)

The tunnels behave like line sinks and the inflow rate is assumed constant for a duration of \( \tau \) (Murdoch et al. 2012). The Jacob (1940) log approximation can be integrated over a transient line source to determine the space-time average hydraulic head change, \( \Delta h \), in the formation at \( t=\tau \) (Murdoch et al. 2012).

\[
\Delta h \approx \frac{\ln(2.25) \Delta Q'}{4\pi} \frac{\Delta Q'}{K_{avg}}
\]

(1.9)
where $\Delta Q'$ is the change in flow rate per unit length of the tunnel. Solving for $\Delta Q'$ in Eq. (1.9) and normalizing by the volume of the formation per unit length of each tunnel, $L^2$, gives $C_g = 4\pi/\ln(2.25) \approx 15.6$ in Eq. (1.6) (Murdoch et al. 2012).

**Boundary Conditions**

Boundary conditions for the mechanical analysis include a free-surface boundary along the top surface of the model geometry and roller boundaries along all four sides and bottom of the model (Fig. 1-21). These boundary conditions were chosen under the assumption that at the sides and bottom boundaries motion can occur in the boundary plane but not perpendicular to the boundaries.

The hydrologic boundaries include zero flux (no-flow) boundaries along the sides and bottoms of the model with a constant recharge rate evenly distributed along the top surface of the model (Fig. 1-21). The rate of recharge (0.15 m/yr) was assumed to equal the baseflow rate estimated from the surface hydrologic budget performed on the Whitetail Creek gauging station data. The streams were considered constant head boundaries with the heads along these boundaries defined by the DEM elevation file used...
to delineate the stream channels. The surface of the tailings pond was also considered a constant head boundary with the head defined using the same DEM data (Fig. 1-21).

**Material Properties**

The initial rock properties in the analysis were taken from studies done in the mine, regional studies of the area, and from evaluations of similar rock types, Table (1.1). The hydraulic and mechanical properties can vary at multiple scales, but it is assumed that at the scale of the analysis they are dominated by layering and stress. Orientations of the layering and foliation (~N20W-60NE) are fairly consistent throughout the region, and some faults and fractures follow this trend (Caddey et al. 1990, Rahn and Roggenthen 2002, Murdoch et al. 2012). The Young’s and Shear moduli, $E$ and $G$, and hydraulic conductivity, $K$, are assumed to be anisotropic, with principle values that align with foliation. Fractures parallel to layering are assumed to control the effective bulk mechanical and hydraulic properties, and it is the deformation of the fractures that causes the properties to vary as a function of effective stress. It will be assumed that properties vary as a function of the vertical effective stress. As a result, the initial properties are assumed to vary with depth, but they are uniform and anisotropic at any given depth.
### Table 1.1 Estimates of hydrogeologic properties in the vicinity of the former Homestake mine. Modified from Murdoch et al. (2012).

<table>
<thead>
<tr>
<th>Property</th>
<th>Value</th>
<th>Method</th>
<th>Source</th>
</tr>
</thead>
<tbody>
<tr>
<td>Hydraulic conductivity</td>
<td>$10^{-9}$ m/s</td>
<td>Pump test in mine</td>
<td>Zhan (2002)</td>
</tr>
<tr>
<td>Porosity</td>
<td>0.01</td>
<td>Estimate</td>
<td>Davis et al. (2009)</td>
</tr>
<tr>
<td>Specific storage</td>
<td>$10^{-7}$ m$^{-1}$</td>
<td>General value for fractured crystalline rock</td>
<td>Wang (2000), Domenico and Mifflin (1965)</td>
</tr>
<tr>
<td>Young’s Modulus</td>
<td>90$^a$, 59$^b$, 77$^c$ GPa</td>
<td>Avg. values from lab tests</td>
<td>Pariseau (1985)</td>
</tr>
<tr>
<td>Shear Modulus</td>
<td>30$^d$, 28$^e$, 48$^f$ GPa</td>
<td>Avg. values from lab tests</td>
<td>Pariseau (1985)</td>
</tr>
<tr>
<td>Poisson’s ratio</td>
<td>0.19$^d$, 0.17$^e$, 0.19$^f$</td>
<td>Lab studies on core from mine</td>
<td>Pariseau (1985)</td>
</tr>
</tbody>
</table>

$^a$ Parallel to dip (x’)
$^b$ Normal to foliation (z’)
$^c$ Parallel to strike (y’)
$^d$ In the x’ and z’ plane
$^e$ In the z’ and y’ plane
$^f$ In the x’ and y’ plane

The hydraulic conductivity, $K$, specific storage coefficient, $S_s$, and effective porosity, $n_e$, are assumed to be related to the fracture density, $\psi$, and the average hydraulic aperture, $\delta$ (Murdoch et al. 2012). The fracture density is the average number of fractures per characteristic length and is taken from Murdoch et al. (2012), equaling 28 m$^{-1}$. The average hydraulic conductivity of a set of parallel fractures is given by Bear (1979)

$$K_p = \frac{\psi \delta^3 \gamma}{12 \mu}$$  \hspace{1cm} (1.10)

where $\psi$ is the fracture density, $\delta$ is the average hydraulic aperture, $\gamma$ is the fluid unit weight, and $\mu$ is the dynamic viscosity. The hydraulic conductivity normal to the parallel fractures is dominated by the matrix conductivity. Studies by Rahn and Johnson (2002)
suggested that $K_{\text{max}}/K_{\text{min}} \approx 5$ at a nearby site underlain by foliated metamorphic rocks similar to those encountered at Homestake. The hydraulic conductivity tensor used in the analysis was taken from Murdoch et al. (2012) and had three principle components. There are two components that are horizontal and parallel to the strike and dip directions and one that is vertical. The horizontal component parallel to strike ($K_x'$) and the vertical component ($K_z'$) are given by Eq. (1.10) and the horizontal component parallel to the dip direction ($K_y'$) is defined as $K_y' = K/5$.

The effective porosity is taken from Murdoch et al. (2012) as

$$n_e = \psi \delta$$

(1.11)

The numerical analysis assumes that the storage coefficient is the unconstrained specific storage, $S_\sigma$, calculated as

$$S = S_\sigma = \gamma \left( \frac{1}{K_b} - \psi \delta \beta \right)$$

(1.12)

where $\beta$ is the compressibility of water and $K_b$ is the bulk modulus of the fractured rock. It is implied that changes in the principle stresses due to change in fluid pressure are independent of orientation ($\Delta \sigma_{xx} \approx \Delta \sigma_{yy} \approx \Delta \sigma_{zz}$) (Murdoch et al. 2012). The bulk modulus includes effects from both the solid rock and the fractures and is given by Eq. (1.13) (Murdoch et al. 2012).

$$\frac{1}{K_b} = \frac{1}{K_s} + C_n \psi' = \frac{3(1-2\nu)}{E_s} + C_n \psi'$$

(1.13)

where $K_s$ and $E_s$ are the bulk modulus and Young’s modulus of the solid rock and $C_n$ is the normal fracture compliance (Murdoch et al. 2012).
\[ C_n = \frac{\partial \delta}{\partial \sigma_e} \] (1.14)

where \( \sigma_e \) is the effective stress. Eq. (1.13) assumes that the fractures are randomly oriented and the interaction between fractures is ignored (Germanovich and Dyskin 1994, Murdoch et al. 2012). When calculating the bulk modulus used in Eq. (1.12), the average Young’s modulus from the three values in Table 1.1 is used.

In the stress-strain portion of the numerical analysis, the fully anisotropic Young’s and Shear moduli from Table 1.1 are used to define the properties of the rock and are oriented with the layering of the region. Using this method the solids in the numerical analysis behave like they have fully anisotropic moduli.

**Changes in Properties with Depth**

The analysis assumes that the hydraulic and mechanical properties of the rock are a function of the effective vertical stress and change with depth which results from opening of the fractures with decreasing depth due to the reduction of vertical stress with decreasing depth. One method for simulating this effect is by assuming a relationship between the effective fracture aperture and the effective stress acting on the fracture. Relationships have been presented by several including Jaeger et al. (2007), Bai and Elsworth (2000), and Ruqvist and Stephansson (2003) describing such relationships. According to Ruqvist and Stephansson (2003), the most common of these methods is given by Eq. (1.15)

\[ \delta = \delta_o + \frac{\left( \delta_o - \delta_{\min} \right) C_n \sigma_e}{\delta_o - \delta_{\min} - C_n \sigma_e} \] (1.15)
where $\delta_o$ is the contact aperture, $\delta_{\text{min}}$ is the minimum aperture under infinite stress, and $C_{ni}$ is the initial normal compliance of the fracture or the normal compliance at zero effective stress (Bandis and Barton 1983). Substituting Eq. (1.15) into Eq. (1.14) gives the compliance of a fracture as

$$C_n = \frac{C_{ni}(\delta_o - \delta_{\text{min}})^2}{(\delta_o - \delta_{\text{min}} - C_{ni}\sigma_e)^2} \quad (1.16)$$

and the effective stress is given by

$$\sigma_e = \sigma_T + \alpha_f P \quad (1.17)$$

where $\sigma_T$ is the total stress with tensile stresses being positive and compressive stresses being negative, $P$ is the fluid pressure in the pores, and $\alpha_f$ for a single fracture is the ratio of open area to the total area of the fracture surface (Murdoch and Germanovich 2006). Typically $\alpha_f$ depends on the characteristics of the individual fractures, fracture density, and the geometry of the fracture network in a fracture medium (Murdoch et al. 2012). Because not all of these parameters are known and to simplify the analysis and reduce the number of parameters, $\alpha_f$ is assumed to equal $\alpha$ (Murdoch et al. 2012).

The initial vertical effective stress prior to mining activities is given by

$$\sigma_{ei} = -(\gamma_T - \gamma_w)d \quad (1.18)$$

where $\gamma_T$ and $\gamma_w$ are, respectively, the unit weights of the rock and water, and $d$ is the depth below the ground surface. It is assumed that the vertical stress is due only to gravity and the horizontal stresses result from material properties and residual tectonic stresses. The initial distribution of the macroscopic hydrogeologic properties with depth
is obtained by substituting Eq. (1.18) into Eqs. (1.10)-(1.16). In the hydrologic analysis it is assumed that $\sigma_T$, from Eq. (1.17) does not change and that the changes in effective stress result from changes in pore pressure.

**Poroelastic Properties**

The parameters for the poroelastic simulation were taken from Murdoch et al. (2012) and include the drained bulk Young’s modulus, $E_b$; drained Poisson’s ratio, $\nu$; hydraulic conductivity, $K$; Biot-Willis coefficient, $\alpha$; and the Biot modulus, $M$. The drained bulk Young’s modulus is given by Eq. (1.19)

$$E_b = E_s \left( \frac{1}{1 + \theta} \right)$$

(1.19)

where $\theta$ is given by

$$\theta = \frac{E_s C_n \psi}{3(1-2\nu)}$$

(1.20)

Neglecting the difference between the Poisson’s ratios of the fractured rock and the solid rock gives $\alpha$, as

$$\alpha = 1 - \frac{K_b}{K_s} \approx 1 - \frac{E_b}{E_s} = \frac{\theta}{1 + \theta}$$

(1.21)

In the poroelastic analysis the storage coefficient in Eq. (1.1) was presented in Wang (2000)

$$S_\varepsilon = \frac{1}{M} = S_\sigma - \frac{3\alpha^2 (1-2\nu)}{E_b}$$

(1.22)

and $S_\sigma$ in this Eq. is given by Eqs. (1.12), (1.13), (1.16), and (1.17) with $\alpha_f = \alpha$. 
The effective stress, $\sigma_e$, in the poroelastic analysis is found by adding the stress found with Eq. (1.2) and the elastic constitutive relationships (Bai and Elsworth 2000) to Eq. (1.18).

**Properties in the Mine Region**

The effective Young’s modulus decreases and the Biot-Willis coefficient increases as a result of mining activities. This occurs when the effects of the tunnels and stopes created during mining are scaled up to the size of the mesh. This was done using

$$
E = E_b \begin{cases} 
\text{intact rock} \\
\xi E_b \text{ mined out region}
\end{cases}
$$

(1.23)

where $\xi$ was determined by calibration using the measured in-situ stresses.

In the mined-out regions the Biot-Willis coefficient is assumed to be 1.0 as taken from Murdoch et al. (2012), whereas it is generally much less than this in the pristine rock at depth.

The approach described above allows the nine macroscopic properties used in the hydrologic analysis ($K, S_s, n_e$) and the poroelastic analysis ($K_b, M, E_b, \nu, \alpha, \xi$) to be defined by four parameters describing the fracture system ($\delta_o, \delta_{\min}, C_n, \Psi$), two defining the rock matrix ($\nu, E_o$) and one defining the mined-out regions, $\xi$ (Murdoch et al. 2012).

Of these parameters, five ($\nu, E_o, \delta_o, \delta_{\min}, C_n$, $\xi$) are reasonably well constrained through the calibration process. This method is similar to the concept of effective properties of heterogeneous (fractured) materials and provides a reliable basis for defining
macroscopic properties that reduces the degrees of freedom in the simulations (Germanovich and Dyskin 1994, Murdoch et al. 2012).
RESULTS OF NUMERICAL ANALYSIS

The numerical analysis was performed in several steps. First the coupled hydraulic and mechanical analyses were run under steady state conditions with the mine omitted to estimate initial conditions. The analysis was then run under transient conditions with the effects of the mine included. Calibration was performed by comparing the results from the transient analyses to known or estimated water levels, flow rates and in-situ stress measurements. The parameters were adjusted until the results best matched the known data. The adjusted hydraulic parameters included the initial fracture aperture, $\delta_o$, the minimum fracture apertures $\delta_{\text{min}}$ and the initial normal compliance of the fracture, $C_{nl}$. The mechanical calibration parameters included the density of the rock, $\rho_{\text{rock}}$, Poisson’s ratio, $\nu$, and the ratio of effective Young’s modulus in the mined region to the ambient Young’s modulus, $\xi$.

The strike of the regional foliation was also adjusted until the predicted principle stress orientations best fit in-situ measurements. This was done after the hydraulic and mechanical calibration sequences were completed and involved holding all other factors constant while varying the strike angle until the orientation of the predicted stresses fell within the range found in-situ.

Calibration

The calibration process involved manually searching the parameter space using a sequential approach. Automated parameter estimation procedures were impractical because of the long run times required for each simulation. Initial values for the
calibrated hydraulic properties were estimated from a previous study by Murdoch et al. (2012) and were initially $\delta_o=30 \, \mu m$, $\delta_{\text{min}} = 3.8 \, \mu m$, and $C_{ni} = 1 \times 10^{-10}$. Initial values for the mechanical parameters were taken from Murdoch et al. (2012), available data (Table 1-1), and by analogy to similar settings, and were initially $\nu=0.2$, $E_{\parallel \text{dip}} = 90 \, \text{[GPa]}$, $E_{\parallel \text{strike}} = 77 \, \text{[GPa]}$, $E_{\perp \text{layering}} = 59 \, \text{[GPa]}$, $\rho_{\text{rock}} = 2500 \, \text{[kg/m}^3\text{]}$, $\xi = 1\%$, and it was assumed that $\psi = 28 \, \text{[m}^{-1}\text{]}$.

**Hydraulic Calibration**

The hydraulic calibration was performed by manually adjusting $\delta_o$, $\delta_{\text{min}}$, and $C_{ni}$ until the predicted flow rate into the mine was within 20 percent of measurements of flow rate from the mine (Fig. 1-22). The flow rate out of the mine was estimated by integrating $R$ over the volumes of the ore bodies. To calibrate the hydrology first the initial normal compliance, $C_{ni}$, and the initial fracture aperture, $\delta_o$ were held constant and the minimum fracture aperture, $\delta_{\text{min}}$, was adjusted until the predicted flow rate best fit measured flow rates. Then $\delta_{\text{min}}$ and $C_{ni}$ were held constant and $\delta_o$ was adjusted to improve the estimation. Finally $\delta_o$ and $\delta_{\text{min}}$ were held constant and $C_{ni}$ was adjusted to
further improve the predicted flow rate. This process was repeated until the hydraulic parameters gave the best fit to the predicted flow rate. Hydraulic calibration was performed before mechanical calibration and then again after mechanical calibration to adjust for any changes to the flowrate that occurred because of changes to the mechanical properties.

Calibration resulted in estimates of $\delta_o = 37$ [µm], $\delta_{\text{min}} = 3.7$ [µm], and $C_{ni} = 1.1 \times 10^{-10}$ [m/Pa] which gives a total flow rate into the mine workings of 0.046 [m³/s] in the early 1990’s. Zhan (2002) and Zhan and Duex (2010) estimated that naturally occurring flow into the mine during this same period was approximately 0.042 to 0.048 [m³/s].

The initial fracture aperture is 7 µm greater, the minimum fracture aperture is 0.1 µm smaller, and the fracture compliance is $10^{-11}$ m/Pa larger than estimates given by Murdoch et al. (2012). The difference between the findings by Murdoch et al. (2012) and this study are due to changes in the model geometry and volume of the workings region as well as inclusion of anisotropic conditions and mechanical properties.

The hydraulic head at the surface of the model was compared to water levels measured from around the Homestake mine as a check of the calibration results. Typically, the predicted head from the simulation was lower than the measured heads with an average residual of approximately -100 m. The topographic relief around the mine is several hundred meters, but the model assumes a uniform surface elevation of 1570 m. Hydraulic head is likely elevated above 1570 m in upland areas between the streams where the shallow permeable zone predicted by Eq. (1.10 and 1.15) should be at
a higher elevation than 1570m. Ignoring topography explains why the model underestimates observed heads by approximately 100m. This is considered an acceptable error between predicted and measured heads at the surface because the models main purpose was to look at pressure changes at depth around the mine.

**Mechanical Calibration**

The mechanical analysis was calibrated by calculating principle stresses at points in the model domain that correspond to locations where in-situ stress data had been measured (Fig. 1-23) (Johnson et al. 1993, Girard et al. 1997, and Golder Assoc. 2010). Stress data presented by Pariseau (1985) were also considered during calibration as a check of the models’ findings. The in-situ stress measurements from Johnson et al. (1993), Girard et al. (1997), and Golder Assoc. (2010) were made by overcoring with soft inclusion cells, as described by Amadei and Stephansson (1997) and Ljunggren et al. (2003). The residual between simulated and measured values was minimized by adjusting $\rho_{\text{rock}}$, $\nu$, and $\xi$, and including initial stresses. It was assumed that the vertical stress is due solely to gravitational loading, so the first step in the calibration process was to hold $\nu$ and $\xi$ constant and vary $\rho_{\text{rock}}$. This gave a density of approximately 2900 kg/m$^3$ and a vertical stress gradient of approximately -0.028 MPa/m. Once the density of the
rock had been constrained, $\rho_{\text{rock}}$ and $\xi$, where held constant and $\nu$ was adjusted until the predicted principle stress obtained from measurements outside of the ore body regions best fit the principle stress estimated from in-situ data from outside the ore regions. $\rho_{\text{rock}}$ and $\nu$ were then held constant, while $\xi$ was varied to match the observed stresses in the vicinity of the mined-out regions. The process was repeated by sequentially varying $\nu$ and $\xi$ until the best fit between the predicted and in-situ stresses was found.

The overall parameter estimation method yielded average properties of $\rho_{\text{rock}} = 2900$ kg/m$^3$, $\nu = 0.32$, and $\xi = 0.31$. The residual stresses found

Figure 1-23. Predicted Vertical, 2$^{\text{nd}}$, and 3$^{\text{rd}}$ principle stress compared to measured in-situ values. Black red circles' = In-Situ Data with error bars; Yellow triangles = Predicted Data. Girard et al. (1997) site at elevation = -652 m.
during calibration can be expressed with Eq. (1.24)

\[ \sigma_r = a_1 + a_2 z \]  

(1.24)

where \( z \) is the depth below ground surface and \( a_1 \) and \( a_2 \) for stresses parallel to strike are -19 MPa and -0.01 MPa/m, respectively. For stresses perpendicular to strike, \( a_1 \) and \( a_2 \) are -15 MPa and -0.009 MPa/m, respectively.

Uncertainty in the field measurements of in-situ stress was included in evaluation of the parameter values. None of the available data on in-situ stress measurements includes estimates of uncertainty, so the general uncertainty resulting from the overcoring method was evaluated. Amadei and Stephansson (1997) reviewed several overcoring studies and estimated that the expected uncertainty is at least 10-20% of the total measurement. Amadei and Stephansson (1997) suggested that this error can be considerably larger when anisotropic conditions are not accounted for and the error may be over 100%. The in-situ measurements at Homestake did not include anisotropy, however Amadei (1983) recommended that below a critical moduli ratio of \( E_{\text{max}}/E_{\text{min}} = 2 \) anisotropic conditions can be reasonably ignored. \( E_{\text{max}}/E_{\text{min}} \) near Homestake is roughly 1.5 according to Table 1.1, therefore, it is reasonable to not account for anisotropy when analyzing the in-situ overcoring data. A recent study by Sjöberg et al. (2003) found that the Borre overcoring method had an absolute imprecision of at least 1-2 MPa and an additional relative imprecision of at least 10%. The Borre overcoring method is similar to the CSIRO method used by investigators at the Homestake mine, with the major difference being the Borre method uses a datalogger which permits continuous measurement of the strain gauges before, during and after the overcoring process.
(Ljunggren et al. 2003). The stress data at Homestake result from measurements by a variety of investigators over more than 20 years, and it seems reasonable to expect that the level of uncertainty is at least ±15% based on the results of Amadei and Stephansson (1997) and Sjöberg et al. (2003), and it could be considerably larger.

The stress data derived from the calibrated model typically matches the in-situ stress measurements when a ±15% uncertainty in the in-situ data is included (Fig. 1-23). An exception is that vertical stress data presented by Girard et al. (1997) differ significantly from other measurements at the same level and from the model results (Fig. 1-23). The predicted vertical compressive stress at the Girard et al. (1997) site is roughly 2.5 MPa greater than the lowest bound of the in-situ measurement as found with the 15% uncertainty (Fig. 1-23). The Girard site in the model is located at an elevation of -652 m in the region of solid rock between the East ore body and the 13&15 ledges and West Ore Body. This region experiences an increase in stress because of softening of the mine workings, which transfers the compressive gravity load to the solid rock around the workings. This effect is one reason that the predicted stress at this location is greater than the measured in-situ values and accounts for roughly 1.5 MPa of variation. Another cause of the large difference between the measured and in-situ vertical stress at the Girard et al. (1997) site could be natural variation in the in-situ stress values. Repeat overcoring measurements at the same site can vary by up to 20 MPa (Johnson et al. 1993 and Golder Assoc. 2010). At the Girard site only one measurement was taken however, therefore the possible variability at this site cannot be quantified. It seems feasible that the large
difference between the stress measurement and the predicted data at the Girard site is because the measurement is affected by either error or local variability.

Orientation of the ambient stresses is assumed to be uniform over the region, with a maximum compression direction that is vertical. Orientation of the minimum principle compressive stress that best fits the data is N50E, and the orientation of the intermediate stress is N40W. These results indicate that the minimum compression is roughly normal to the strike of layering, whereas the intermediate values are roughly parallel to strike. These orientation data were obtained as a best fit to the magnitudes of the measured stresses, and it is important to point out that the measured stress orientations were not used in the parameter estimation process. As a result, the measured orientation data serve as a check to the results of the parameter estimation process (Fig. 1-24). The best fit for the in-situ stress data is a trend of N40W (Fig. 1-24). The observed orientation data are highly variable, and in-situ measurements a few meters apart can differ by up to 90 degrees (Johnson et al. 1993, Girard et al. 1997, Golder Assoc. 2010). The estimated orientation for the in-situ stresses appears to be a reasonable approximation to the measurements (Fig. 1-24).
Figure 1-24. Intermediate and minor (least compressive) principle stress orientations and magnitudes, Pariseau (1985), Johnson et al. (1993), Girard et al. (1997), and Golder Assoc. (2010). Radial axis is stress magnitude [MPa], polar axis is orientation [deg.]. Dashed lines are results from the model, which assumes the orientation is homogeneous.

**Error in Mechanical Parameters**

Estimates for the parameters $\rho_{\text{rock}}$, $\nu$, and $\xi$ were bounded by determining the range where the predicted stresses were within ±15% (the uncertainty of the stress data) of the field data. The density of the rock, $\rho_{\text{rock}}$, has the greatest effect on the vertical stress, therefore the vertical stresses were used to bound $\rho_{\text{rock}}$. This was first done by determining the maximum and minimum vertical stress gradients that are within ±15% of the vertical in-situ stresses. This method gave a fairly broad range, $1900 \text{ kg/m}^3 \leq \rho_{\text{rock}} \leq 7400 \text{ kg/m}^3$, the upper bound of which is much higher than typical densities of similar rock types (Smithson 1971). Another method considered was to bound $\rho_{\text{rock}}$ directly with the estimated uncertainty. This method assumed that because the vertical stress is
proportional to the rock density and because the error in the stresses is assumed to be ±15%, then the error in the density can also be assumed to be ±15%, giving bounds of $2400 \text{ kg/m}^3 < \rho_{\text{rock}} < 3300 \text{ kg/m}^3$. These bounds are similar to the range of densities found by Smithson (1971) from numerous metamorphic terrains which ranged from 2580 kg/m$^3$ to 3260 kg/m$^3$.

![Vertical Stress with Depth](image)

Figure 1-25. Vertical stress data at the three in-situ sites with vertical stress gradients. Black line is the vertical stress gradient from calibration.

![Diagram of Stress vs. Poisson's Ratio](image)

Figure 1-26. (a) Intermediate and least principle stresses as functions of Poisson’s ratio at the Girard site; and (b) the least principle stress as a function of the ratio between the effective Young’s modulus of the ore bodies and the Young’s modulus of the host rock ($\zeta$) at three sites.
The Poisson’s ratio, $\nu$, and the ratio of effective Young’s modulus in the mined region to the ambient Young’s modulus, $\xi$, were also bounded using the ± 15% error range for the data. This was done by calculating the stresses as functions of the parameters at the different measurement locations. The results were plotted for each site and the slopes $d\sigma / dv$ and $d\sigma / d\xi$ were determined in the range of the best-fit parameter values (Fig. 1-26). Then the upper and lower bounds of the in-situ stress data were calculated using the ± 15% uncertainty. From this the change in $\nu$ and $\xi$ needed to exceed these bounds was calculated as

$$\Delta\nu_{\text{lower}} = \frac{(\sigma_{\text{L.bound}} - \sigma_{\text{predicted}})}{d\sigma/d\nu}$$

(1.25)

$$\Delta\nu_{\text{upper}} = \frac{(\sigma_{\text{U.bound}} - \sigma_{\text{predicted}})}{d\sigma/d\nu}$$

(1.26)

$$\Delta\xi_{\text{lower}} = \frac{(\sigma_{\text{L.bound}} - \sigma_{\text{predicted}})}{d\sigma/d\xi}$$

(1.27)

$$\Delta\xi_{\text{upper}} = \frac{(\sigma_{\text{U.bound}} - \sigma_{\text{predicted}})}{d\sigma/d\xi}$$

(1.28)

where, $\sigma_{\text{L.bound}}$ and $\sigma_{\text{U.bound}}$ are the lower and upper uncertainty bounds for the stresses respectively and $\sigma_{\text{predicted}}$ are the respective predicted stresses from the simulation. These equations gave the increase and decrease in the parameters $\nu$ ($\Delta\nu_{\text{lower}}$ and $\Delta\nu_{\text{upper}}$) and $\xi$ ($\Delta\xi_{\text{lower}}$ and $\Delta\xi_{\text{upper}}$) needed to reach the lower and upper stress error bounds. Then the
average increase and decrease in ν and ξ needed to reach the bounds were calculated and added to the calibrated values of ν and ξ to determine their upper and lower bounds as based off of the ± 15% stress error.

When bounding ν this method was applied to the horizontal principle stresses for all sites where in-situ stresses were measured, and when bounding ξ it was applied to all three principle stresses at the Johnson et al. (1997) site. The reason for this selectivity is that change in ν has a minor effect on the vertical stresses (less than 1/40 of the effect on the horizontal stresses), so they were ignored. Furthermore the parameter ξ has a significant effect on the predicted stresses within and in close proximity to the ore bodies, but it has minimal effect outside of these regions.

The results of the analyses of Poisson’s ratio varied among the sites, with the greatest sensitivity at the Girard site, where \( d\sigma/d\nu = -71 \text{ MPa} \) and \(-80 \text{ MPa}\) for the 3\(^{rd}\) and 2\(^{nd}\) principle stresses, respectively. The average change in Poisson’s ratio to exceed the upper and lower stress bounds was approximately ±0.1, and from that it follows that 0.22 < ν < 0.42. For comparison, Golder Assoc. (2010) measured multiple locations on the 4850 L and found that Poisson’s ratio ranged from 0.25 to 0.36 with a mean of 0.30, a result that is essentially the same as the calculated values.

The least principle stress at the Johnson et al. (1997) site is a non-linear function of ξ over the full range of ξ \((0 < \xi < 1)\), but the slope is approximately -0.28 MPa in the range of the observed stresses (Fig. 1-26). This gives an average uncertainty in the parameter ξ of approximately ±0.16, which gives a range of 0.15 < ξ < 0.47.
<table>
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<th>Min</th>
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<td>0.22</td>
</tr>
<tr>
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<td>2400</td>
<td>3300</td>
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Distribution of Properties with Depth

The constants obtained from calibration were used along with equations provided above to estimate the distribution of hydrogeologic properties. The properties depend on fracture aperture, which is sensitive to stress, and this causes the properties to vary as a function of depth. The change in the parameters is greatest in the first few hundred meters below ground surface. The rate of change of the hydraulic aperture diminishes below approximately 200 m, and more than 1000 m below ground surface the aperture changes by only a few tenths of a

<table>
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<th>δ [m]; K [m/s]; Ss [1/m]; ne</th>
<th>δ [m]; K [m/s]; Ss [1/m]; ne</th>
<th>δ [m]; K [m/s]; Ss [1/m]; ne</th>
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</table>

Figure 1-27. Plots showing the average hydraulic aperture (δ), hydraulic aperture (δ), hydraulic conductivity (K), specific storage (Ss), effective porosity (ne) and Young's modulus (E) and Biot-Willis coefficient (α) as functions of depth.
micron over several hundred meters. As a result, the hydraulic and mechanical parameters are nearly independent of depth below 1000 m (Fig. 1-27).

The hydraulic conductivity decreases by two orders of magnitude from $10^{-6}$ m/s to $10^{-8}$ m/s in the upper 100 m (Fig. 1-27B). This is consistent with an active shallow groundwater flow system underlain by a deeper system where water fluxes are small (Murdoch et al. 2012). The values for the hydraulic conductivity at depth are similar to the estimates given in Table (1.1). The effective porosity, approximately $1 \times 10^{-3}$ to $1 \times 10^{-4}$, is roughly two orders of magnitude lower than the estimate for porosity in Table (1.1), but this seems feasible because the measured porosities are on samples that have been unloaded. The average Young’s modulus increases with depth, and below 1500 meters it ranges from 75-80 GPa, which is similar to values measured in the lab (Table 1.1).

The distributions of parameters with depth are similar to those described by Murdoch et al. (2012), however, there are some noteworthy differences. Hydraulic conductivity is slightly greater at shallow depth, approximately $1 \times 10^{-6}$ m/s, but decreases and is the same as the values determined by Murdoch et al. (2012) at depth, approximately $1 \times 10^{-8}$ to $1 \times 10^{-9}$ m/s. The specific storage is roughly one order of magnitude greater, approximately $1 \times 10^{-4}$ m$^{-1}$, than that found by Murdoch et al. (2012) at shallow depths but becomes roughly the same below a depth of 1000 m. The distribution of the Young’s Modulus in the previous study by Murdoch et al. (2012) changes more abruptly at depths above roughly 1000 m, and after roughly 1000 m there is little to no change in the Modulus magnitude. However, in this study the variation in Modulus is more gradual above a depth of 1000 m and there is roughly 5 GPa change below this.
depth over roughly 1000 to 1500 m. Below these regions where $E$ is changing in the two models, however, the values for $E$ are approximately the same. In the previous simulation $\alpha$ varied more linearly with depth. However, in the new simulation the distribution approximately mirrors the shape of the Young’s Modulus ($E$) distribution and ranges from 1 at the surface to roughly $1 \times 10^{-2}$ at the base of the mine.

### Hydraulic Head in the Mine Region

Hydraulic head prior to mining slopes to the northeast with local variations associated with streams. The regional hydraulic head gradient is approximately 0.03, and the vector azimuth is SW.

Local variations in hydraulic head are controlled by the streams, and flow vectors have components toward the stream and parallel to the flow direction of the nearest stream, a pattern typical of a local groundwater flow system. In regions above the mine the hydraulic heads before mining typically range from 1700 m to

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**Figure 1-28.** Hydraulic head [m] before mining begins in the vicinity of the mine workings. (a) Cross-section through the East ore body (looking NE). (b) Map view of hydraulic head at ground surface with arrows showing flow directions.
1450 m with the greatest heads above the SE portions of the mine and the lowest along Whitewood creek (Fig. 1-28).

Dewatering causes the hydraulic head to drop roughly 30 to 40 meters above the mine workings (Fig. 1-29). This occurs primarily above the open pit and upper workings region of the mine. Outside of this region, the heads vary little during mining.

The greatest drop in hydraulic head (drawdown) occurs at depth around the mine workings. The drawdown follows the base of the mine downward and expands outward from the workings (Fig. 1-30 & 1-31). The greatest drawdown is roughly 2400 m, and this

Figure 1-29. Hydraulic head [m] and the change in hydraulic head [m] at the surface after the mine is fully dewatered, with arrows showing flow gradient direction. (a) Magnitude of the hydraulic head, (b) Head change because of mining.

occurs at the bottom of the mine (-830 m) when
the mine is fully dewatered (Fig. 1-30). The shape of the region of reduced head is controlled by the geometry of the mine workings and is roughly elliptical in both plan-view (Fig. 1-30) and cross-section (Fig. 1-31). The region of hydraulic head affected by dewatering extends up to several kilometers radially outward from the mine workings. It was assumed at depth that the regions where heads have been definitely altered by mining are regions where at least 100 m of head change have occurred.
Figure 1-31. Change in hydraulic head [m] along a cross-section through the East ore body, looking NE. (a) 20 yrs, (b) 40 yrs, (c) 60 yrs, (d) 100 yrs, (E) when the mine is fully dewatered.
The lateral extent of this region is generally smaller than the vertical extent. Laterally the region extends out roughly 2-3 km at its maximum, which occurs in the direction of the major axis of the mine (Fig. 1-30). The region extends upward and downward from the mine workings. The upward extent is over 1.5 km above the SE portions of the mine. The downward extent of the region where heads have been affected can reach over 4 km. At 4 to 5 kilometers downward from the SE portion of the mine workings the hydraulic head is lowered by up to 100 m. This creates a region of reduced head around the mine workings that resembles an asymmetric teardrop that extends up to 4 kilometers downward from the workings at the lower levels of the mine (Fig. 1-31). The extent of the affected region was considered to be the region where head change was at least 100 m, which seems a reasonable assumption of error for the numerical simulation.

The mine was allowed to refill resulting in an increase in the hydraulic head in the workings by up to 1.0 kilometer. As the workings refill and the head increases, a wave of increased head moves outward into the surrounding rock. The front of increased head extends a few hundred meters into the enveloping rock before dewatering recommences and its outward motion is arrested. Originally it was planned to fully dewater the Homestake mine, but current plans are to dewater the mine to a depth of 1615 m (elevation ~ -45 m) (Short-Elliott-Hendrickson Inc. 2007). The simulation was created before knowledge of this decision had been found and assumes the mine is completely dewatered; therefore it overestimates the amount of dewatering that occurs.
**Hydraulic Head Gradients**

The hydraulic gradients vary because of dewatering. The hydraulic gradient is altered in the vicinity of the mine, and this affected region extends out from the workings about 1 km horizontally and up to 4 kilometers vertically. The magnitude of the gradient increases with proximity to the mine, and gradients are oriented normal to the regions of reduced head in and around the workings.

![Figure 1-32. Vertical hydraulic gradient \( L/L \) along a cross-section through the East ore body (looking NE). (a) @ 60 yrs, (b) @ the end of mining activities.](image)

The magnitudes of the vertical and horizontal components of the gradient are typically similar, although the horizontal components exceed the vertical components along the sides of the major ore bodies at depth (Fig. 1-32 & 1-33). The vertical component dominates above and below the mine workings, and the horizontal components dominate on the sides of the workings.
Figure 1-33. The N-S (a & a’) and E-W (b & b’) hydraulic gradients along a vertical cross-section through the East ore body (looking NE) and along a horizontal cut-plane at an elevation of -30m.
The overall magnitude of the hydraulic gradient, Eq. (1.29), increases with time as mining occurs.

\[
\text{Gradient Magnitude} = \sqrt{\left(\frac{dh}{dz}\right)^2 + \left(\frac{dh}{dx}\right)^2 + \left(\frac{dh}{dy}\right)^2}
\]  

(1.29)

Along the border of the ore regions the magnitude of the hydraulic head gradient ranges from less than 1.0 to over 13 \([L/L]\) in very small regions at the most SE tip of the East and Main ore bodies. At the surface of the model, above an elevation of roughly 1340 m, the magnitude of the hydraulic gradient is less than 1.0 and typically less than 0.2 \([L/L]\). Gradients in the range of 0.2 occur near the streams, but they range from 0.001 up to 0.1 in the upland areas between streams \([L/L]\). In approximately the upper kilometer of the mine (elevations > ~500 m) the

Figure 1-34. The magnitude of the hydraulic head gradient, along the cross-section through the East ore body (looking NE). (a) @ 60 yrs, (b) @ the end of mining
hydraulic head gradient is as great as 6 \([L/L]\), with typical values of approximately 3. The largest gradients above an elevation of approximately 1125 to 1150 m occur at the top of the East and Main ore body regions with gradients above the Main ore body exceeding 3. In the upper kilometer below approximately 1150 m the greatest gradients occur along the SW and NE sides of the working regions. In the upper regions of the workings, the region where gradients are notably affected by mining extends out from the workings laterally to the NE, NW, and SW of the workings, by approximately 400 m and to the SE by up to almost 3 km (Fig. 1-35). In the upper kilometer of the model the hydraulic gradients are altered in the regions of solid rock above the lower portions of the mine. In these regions of solid rock the hydraulic gradient ranges from approximately 1 to 1.75 \([L/L]\) and is typically roughly 1.5 \([L/L]\). Outside of the region, where mining has affected the hydraulic gradients, the gradient ranges from approximately 0.001 to 0.009 \([L/L]\).
Figure 1-35. Magnitude of hydraulic gradient [L/L] when the mine is fully dewatered at various levels in the mine; (a) 1370 m, (b) 1200 m, (c) 1000 m, (d) 600 m, (e) -400, (f) -830 m.
Larger gradients occur in the lower portions of the mine workings (elevations < ~500 m), and typically along the boundaries of the workings the hydraulic gradient ranges from 2 up to 13 \([L/L]\). In general the largest gradients, excluding the small location at the tip of the East and Main ore bodies, occur along the NE side of the East and Main ore bodies between elevations of -500 to -600 m. In this region the hydraulic gradient can exceed 13 \([L/L]\). The typical gradient at the boundary of the workings in the lower region of the mine is 4 to 5 \([L/L]\).

At the base of the mine, less than an elevation of approximately -600 m, the average gradient at the boundary of the workings is generally 6.5 to 7.5 \([L/L]\). Below an elevation of -600 m notable changes in the gradient occur mainly around the West Ore Body and the lower extent of the Main ore body and 13\textsuperscript{th} Ledge, which are the only regions of workings that extend below this elevation. At the lowest levels of the mine, the region where hydraulic gradients are elevated compared to typical background values found in regions unaffected by mining, 0.001 to 0.009 \([L/L]\), extends perpendicularly out from the workings by up to almost 1 km or more in areas to the SW of the West Ore Body and to the SE of the SE tip of the East and Main ore bodies.

**Flow Systems**

There are two primary flow systems that exist around the mine region, a shallow flow system in the upper few hundred meters and an underlying deep flow system. The shallow flow system is dominated primarily by ground water that is recharged in upland areas and discharges to nearby streams or to the tailings pond. Six surface watersheds are in the model (Fig. 1-36).
Figure 1-36. Map of shallow groundwater flowpaths with delineated watersheds for the major streams, Spearfish system (purple), False Bottom (green), Polo/Miller (orange), Whitwood/Whitetail (blue), Two-bit/Bear Butte (pink), and Elk Creek (yellow). Color of flow lines gives flow path travel times [$x10^9$ sec] starting from $t=0$. 
The shallow flow system occurs in the upper 250-300 meters of the model (Fig. 1-37 & 1-38), where the hydraulic conductivity, \( K \), is one to two orders of magnitude larger than \( K \) at depth (Fig. 1-27). Travel times within the shallow flow system are typically short with resident times that range from days to years (Fig. 1-36 & 1-37). The longest travel times in the shallow flow system are on the order of several decades. Flow paths in the shallow system can range from 10’s of meters to over 5 kilometers with a typical distance of around 1 to 2 kilometers (Fig. 1-36 & 1-37). Flow originating near the streams has the shortest travel times and flow paths. Fluid particles with the longest travel times and flow paths originate from regions with the lowest stream densities. There is also flow from the SW to the NE that is caused by regional topography.
Figure 1-37. Hydraulic head [m] in the mine workings and flow paths of fluid from around the mine, starting along a cross-section through the East ore body (looking NE). Left color grid is hydraulic head [m] and right is travel times [x10^9 sec] starting from t=0. (a) @ 20 yrs, (b) @ 40 yrs, (c) @ 60 yrs, (d) @ 80 yrs, (e) when mine is completed and dewatered.
Deep Flow System

Before mining began the deep flow system consisted of slow flows and long resident times, up to many thousands of years. As mining occurs the region of reduced head and changing hydraulic gradients induces flow in the surrounding host rock (Fig. 1-37). Flow caused by dewatering is in general towards the region of lowest hydraulic head within the mine workings. As mining progresses the affected region increases in size and the mine captures water from a larger area. In the first few decades flow to the mine generally originates from within less than one kilometer both laterally and below the workings. Some water originating from within the shallow flow system discharges to the mine workings. When the mine reaches roughly half of its total depth, the region where flow is affected extends laterally outward and vertically below the workings by as much as 2 kilometers (Fig. 1-37). When the mine has reached its maximum depth, the region where flow is affected by mining extends out from the workings by over 5 km below and laterally around the lower portions of the mine (elevations < ~-100 m).

When the mine is flooded, the rate of flow to the mine slows and the rate at which the affected area propagates outward is slowed. The majority of flow to the workings converges on the region of lowest hydraulic head, which approximately follows the level of the water in the mine. Therefore the region in the mine where the greatest amount of flow discharges moves upward with the water level as refilling occurs and then back down during subsequent dewatering.

The deep flow system can be separated into two main fluid capture zones; a Recharge capture zone, and a Storage capture zone (Murdoch et al. 2012). The Recharge
capture zone is the region where water from recharge that has occurred since mining began discharges to the mine workings (Fig. 1-37 & 1-38). The region is approximately elliptical when projected to the surface. Flow to the mine in the Recharge capture zone comes from above the mine and from laterally around the workings with the greatest influx from above the mine. Travel times of fluid in the Recharge capture zone are typically less than 40 years. However, there is flow that comes from recharge that has travel times to the mine greater than this, up to 125 years. This discharge to the mine generally comes from over 2 kilometers away from the mine. Directly around the open pit and upper workings of the mine, in the upper few hundred meters, travel times are on the order of years with path lines on the order of 10’s to 100’s of meters and typically no greater than 1 km (Fig. 1-37).

Figure 1-38. Schematic of the major hydrologic zones which form in the mine region, including the shallow hydrologic system (pink) the upper/recharge capture zone (green) and the lower/storage capture zone (blue). Modified from Ebenhack et al. (2013).
The Storage capture zone is located further out from the workings below and adjacent to the Recharge capture zone and consists of the region where water stored in the rock discharges to the mine workings. The Recharge capture zone is included in this region, but it also extends out laterally and to depth below the Recharge capture zone. Some water in the Storage capture zone originally comes from stored water that was in the recharge zone. This water comes mainly from regions 2 or more kilometers away from the workings, outside of the Recharge capture zone. Water in the Storage capture zone typically has longer travel times and flow paths than water from the upper/recharge capture zone. The longest flow paths in the Storage capture zone can extend to over 4 km below the mine. Travel times in the storage capture zone can reach up to 125 years with typical travel times of approximately 80 years.

There is interaction between the Recharge capture zone and the Storage capture zone. Some water from the Storage capture zone originating from below the NW portion of the upper workings flows upward into the Recharge capture zone before discharging to the upper workings. There is also fluid that originates from within the Recharge capture zone above the SE portion of the mine that flows downward into the Storage capture zone before discharging into the lower workings.

**Fate of Discharge to the Tailings Pond**

During mining and dewatering water that is pumped from the mine region and water used for washing the tailings is discharged to the tailing pond above the south-eastern portion of the mine. There is also discharge to the tailings pond from surface flow. Water from the tailings pond has two main fates, a large portion of the tailings
water discharges to the surface streams around the tailings pond and a smaller amount flows downward and laterally discharges to the mine workings themselves (Fig. 1-39).

The analysis assumes the tailings pond is unlined. Travel times along flow paths from the tailings pond to the streams is typically on the order of years, with a maximum travel time of approximately five years to tributaries in the Whitewood drainage system (Fig. 1-36 & 1-39). Travel times along flow paths to the mine workings are considerably longer and are on the order of several decades with an approximate maximum travel time of 55 years. Water that is discharged into the mine workings ends up in the lower workings over time where the lowest hydraulic head occurs. The tailings pond is likely lined with low permeability material that reduces flow from the pond. Therefore the travel times estimated above are considered to be the worst case scenario of flow from the tailings pond.

Figure 1-39. Flow paths from surface and center elevation of the tailings pond. left color grid is hydraulic head [m], right is time since mining began [x10^9 sec] starting from t=0.
Stresses Before Mining

Stresses in the numerical analysis are compressive, with no tension occurring. The sign convention used for the numerical analysis assumes that compressive stresses are negative and tensional stresses are positive. Using this convention a decrease in stress is an increase in compression, and an increase in stress is a reduction in compression. When presenting the results from the numerical analysis, the first principle (1st) stress is assumed to be the most compressive stress or the most negative stress value. The third principle (3rd) stress is assumed to be the least compressive stress or the least negative stress value. Depths in the numerical analysis are given in terms of elevation with the top of the model at an elevation of 1570 m. With this convention the top of the mine is at 1570 m and the bottom of the mine is at approximately -830 m.

Figure 1-40. Principle stresses with depth before mining occurs (negative stresses are compressive).
The principle stresses before mining are approximately vertical and horizontal. The horizontal principle stresses are aligned with strike and dip direction of the regional layering. The principle stress directions vary with depth, and in approximately the upper one kilometer of the model the vertical stress is the minor (3rd) principle stress and the horizontal stresses are the intermediate (2nd) and major (1st) principle stresses (Fig. 1-40). Below the upper kilometer the magnitude of the vertical stress overtakes the horizontal stresses and the 1st principle stress roughly aligns vertically and the 2nd and 3rd principle stresses are horizontal, Fig. (1-40). The horizontal principle stress directions were controlled by the schistocity in the region with the 2nd Principle stress aligned ~N40W, approximately parallel to strike, and the 3rd Principle stress (least compressive stress) aligned ~N50E, approximately parallel to dip direction, Fig. (1-41). Before mining orientations stay uniform throughout the region both laterally and with depth. Comparing these predicted stresses orientations (Fig. 1-41) to the in-situ orientations (Fig. 1-24), the predicted orientations for both the 2nd and 3rd principle stresses fall approximately along the median orientations found in-situ.

The stresses decrease with depth (become more compressive). The vertical stress gradient is ~ -0.028 MPa/m (Fig. 1-40). This is more than twice the 2nd and 3rd principle stress
gradients, which are \( \sim -0.010 \text{ MPa/m} \) and \( \sim -0.009 \text{ MPa/m} \), respectively (Fig. 1-40). At shallow elevations, above approximately 500 m, the magnitudes of the 2\textsuperscript{nd} and 3\textsuperscript{rd} principle stresses are less (more compressive) than the vertical stress by several megapascals, up to approximately -15 and -19 MPa at the surface, respectively. At elevations of approximately 700 and 400 m, the vertical stress magnitude becomes more negative (more compressive) than the horizontal stress magnitudes. Below these elevations the vertical compressive stress becomes the 1\textsuperscript{st} principle stress and the horizontal compressive stresses become the 2\textsuperscript{nd} and 3\textsuperscript{rd} principle stresses, this corresponds to a depth range of 870 m to 1170 m below ground surface, Fig. (1-40).

The stresses vary with depth because of gravity, and there is a considerably smaller horizontal variation most likely due to anisotropic conditions and fluid pressure variations in the model. The horizontal variation is on the order of 1 to 2 orders of magnitude less than the vertical gradients. The horizontal variation is greatest in a direction approximately aligned with dip direction from SW to NE. In general in the region around the mine, within 10 to 15 km away, the 3\textsuperscript{rd} and 2\textsuperscript{nd} principle stresses increase (become less compressive) from the SW to NE. The vertical stress typically decreases (becomes more compressive) along the SW-NE stress profile. The horizontal gradient magnitudes for all three of the principle stresses are on the order of roughly 0.6 to 1.0 MPa/km.

**Stress Change due to Mining**

Mining changes the magnitudes and orientations of the stresses in the vicinity of the workings. Changes in stress magnitude and orientations are generally confined to
within one kilometer from the workings. The distribution of stress change is controlled by the geometry of the mine workings. In general, the greatest change in stress occurs near the current bottom of the mine. However, there are some regions in the upper portions of the mine where changes in stress can exceed those at depth for individual principle stresses.

**Stress Orientation Change Due to Mining**

Rotations of the principle stress orientations after mining occur along vertical as well as horizontal axes. The change in orientation for the principle stresses can be highly variable and appears to be most affected by depth and relative location and proximity to the ore bodies. The region where stress orientations have been affected can range up to almost 2 km away from the ore bodies to less than 100 m away, with the greatest extent above and below and the least extent laterally around the ore regions. The greatest variations in orientation occur within the East and Main ore bodies where stress rotations can vary by up to 90°. Within the West Ore Body and 13&15 Ledges the rotations are more consistent throughout the ore regions; however, there are variations of 5° to 50°. In these ore bodies, especially the 13&15 Ledges, rotations are most likely also affected by proximity to the East ore body.
In the upper 300 m of the model, rotation of the principle stress directions typically occurs about a vertical axis. Rotation is typically less than at depth and generally ranges from 5° to 30°, Fig. (1-42). In these upper regions rotation occurs in the ore regions, above and typically confined to directly above the ore regions and up to 100 m laterally away from them, notably the East and Main ore bodies. Rotation can be both counterclockwise and clockwise depending on relative location to the ore regions.

In the upper portions of the East and Main ore bodies (elevations > ~800 m) rotation of the principle stress directions is primarily about a horizontal axis that is roughly aligned with strike (± ~5° to 10°). Rotation about this horizontal axis is typically on the order of 30° to 50°. In this region there is a smaller component of counterclockwise rotation about a vertical axis that is on the order of 5° to 10°. In the upper regions of the East and Main ore bodies rotation directions are fairly consistent, and the greatest variations are the magnitudes of rotation which typically vary with depth through the ore regions, with the greatest rotations in the middle of the ore regions. Rotation occurs outside of the upper East and Main ore bodies as well and can extend to almost 2 km horizontally from the workings region. These rotations typically occur.
along horizontal axes that range from parallel to strike to parallel to dip with little to no rotation about the vertical axis. In the regions outside of the workings rotations can reach up to 60° to 70°.

In lower regions of the East and Main ore bodies (elevations < ~800 m) there is increased variation in the stress rotations within the ore bodies. The horizontal axes of rotation in these lower regions can vary by up to almost 90° and the direction of rotation about these axes varies as well. The largest variations occur in the East ore body in locations and depths where the East and Main ore bodies intersect or are laterally next to each other and within the Main ore body itself. In these regions the principle stress orientations at locations a hundred meters or less away from one another can vary by up to 70°, with the greatest variation occurring with changes in depth. In the lower portions of the East and Main workings the change in stress orientations because of mining can reach up to 70° to 80°. Around the lower portions of the East and Main ore bodies the laterally extent away from the workings where stress orientations have been affected is smaller than around the upper portions of the workings. Below an elevation of roughly 300 m, changes in the stress directions because of mining are limited to within 100 to 200 m away from the working regions.

Stress rotation in and around the West Ore Body is mainly confined to within the West ore region with little to no variation up to 100 meters away from the ore body. Stresses within the West Ore Body are less variable than within the East and Main ore regions. Typically the greatest rotations occur in the upper portions of the West Ore Body with reduced rotation in the lower portions. At the base of the West Ore Body
(elevations < ~750 m) there is little to no rotation of the principle stress directions. Typically in the West Ore Body rotation of the principle stresses occur along vertical as well as horizontal axes that are fairly consistent within the ore region; however, there is variation of these axes directions (up to ~30°) near the boundaries of the ore region and in the SE portions of the ore region. Rotation about the vertical axis generally is clockwise with a magnitude that ranges from roughly 15° to 30°. The horizontal axes about which rotation occurs is roughly aligned 5° to 20° clockwise from the strike direction, and rotation along these axes in the upper portions of the region can reach almost 60° to 70°.

Rotations within the 13&15 Ledges are fairly consistent throughout these regions and appear to be affected by their proximity to the West and East ore bodies. In these parts of the workings rotation about the vertical axis can be both clockwise and counterclockwise and typically does not exceed 15° to 20°. Rotation also occurs about horizontal axes that are within 10° to 15° of strike and typically do not exceed 30°.

**Stress Magnitude Changes Due to Mining**

Mining changes the magnitudes of the vertical and two principle horizontal stresses (2nd and 3rd principle stresses) in the vicinity of the mine workings. In the upper portions of the workings and near the surface there is a region where the stresses have been changed by at least 0.5 MPa that extends out from the mine by up to 1 km. The distance away from the mine where stresses have been changed by at least 0.5 MPa decreases with increasing depth, and near the base of the mine it is only 100 to 200 m. The greatest changes in stress occur in the ore bodies and at the boundaries of the
workings and can exceed several 10’s of MPa with typically larger changes at greater depths.

**Vertical Stress Magnitude Changes Due to Mining**

The magnitude of the vertical stress generally increases (becomes less compressive) within, above and below the mine workings and typically decreases (becomes more compressive) adjacent to the workings after mining (Fig. 1-43). There are regions within certain ore bodies and above and below them where there are decreases in vertical stress (increased compression) because of interaction between different ore regions. One case of this is in and above the NE side of the upper East ore

Figure 1-43. Vertical compressive stress change due to mining at elevations of (a) 900 m, (b) 300 m, (c) -300 m, (d) -830 m. Negative values are increases in compression, positive values are decreases.
body where increased compression caused by the Main ore body results in an overall increase in vertical compression (Fig. 1-43).

The greatest increase in vertical stress (decrease in compression), 31 MPa, occurs in the lower few hundred meters of the mine region within the West Ore Body. In the lower portions of the mine stresses are generally increased by 20-30 MPa with the smallest increases within the East and Main ore body regions. As depth decreases the increase in stress within the ore regions diminishes. In the upper portions of the East and Main ore bodies the increase in stress is typically less than 10 MPa (Fig. 1-43). Above and below the mine workings the vertical stress increases by up to 15 MPa with the greatest increase above the lower SE portions of the workings.

The greatest decrease in vertical stress (increase in compression) also occurs in the lower portions of the mine. This occurs in the solid rock regions between the major ore bodies at depth including the West, 13&15, and East ore regions (Fig. 1-43). In these regions the decrease in vertical stress ranges from -10 to -40 MPa. At the SE extreme of the East and Main ore bodies the decrease in vertical stress is nearly -40 MPa. The decrease in vertical stress diminishes with decreased depth and generally ranges from -5 to -15 MPa adjacent to the boundaries of the upper half of the East and Main ore bodies.

2nd Principle Stress Magnitude Changes Due to Mining

The magnitude of the 2nd principle stress typically increases (becomes less compressive) within and above the mine workings and decreases (becomes more compressive) adjacent to the workings (Fig. 1-44 & 1-45). Like the vertical stress, there are regions where interactions between ore regions cause variations from this basic
behavior. One region is above the East ore body in the upper 750 m of the model where a decrease in stress caused by the Main ore body is greater than the increase in stress caused by the East ore body resulting in an overall decrease in stress (Fig. 1-44 & 1-45).

In the upper regions of the workings (elevations > ~820 m) the increase in stress in the

Figure 1-44. 2nd principle stress change [MPa] along a cross-section through the East ore body (looking NE); (a) @ 60 yrs, (b) @ present day. Positive values are increased tension; negative values are increases in compression.

Figure 1-45. 2nd principle stress change [MPa] at present day; (a) Elevation = 1000 [m], (b) Elevation = -400 [m]. Positive values are increases in tension; negative values are increases in compression.
ore regions is as great as 2 MPa but is typically 1 MPa or less. In the lower portions of
the mine (elevations < ~820 m) the increase in 2nd principle stress is greater and typically
increases with depth. The increase in stress in the lower portions of the mine range from
approximately 1 MPa to several 10’s of MPa. The greatest increase in stress occurs at the
base of the West Ore Body and can exceed approximately 30 MPa. Generally the
increase in stress in the lower workings ranges from 5 to 10 MPa (Fig 1-44 & 1-45).
Increases in the 2nd principle stress above the ore regions is greatest at approximately half
of the depth of the mine and can reach 7 to 9 MPa (Fig. 1-44). Typically the increase in
stress above the mine is approximately 1 to 2 MPa.

Significant decreases in the 2nd principle stress occur at two depths laterally
around the workings, near the base of the mine and at approximately 700 m below ground
surface. At these regions the decrease in stress can exceed -13 MPa (Fig. 1-45). In
general, the decrease in 2nd principle stress around the mine ranges from -8 to -10 MPa at
the boundary of the workings. Above the workings in regions where decreased stress
occurs (elevations > ~820 m) the magnitude of decrease is typically -1 to -6 MPa (Fig. 1-
45). In areas above the East ore body to the SW of the Main ore body, the 2nd principle
stress decreases by as much as -13 MPa (Fig. 1-45).
In the upper approximately half of the mine the magnitude of the 3\textsuperscript{rd} principle stress increases (becomes less compressive) in and above the workings region, and there is decreased stress (becomes more compressive) adjacent to the workings. In the lower half of the mine there is increased stress in, above and below the workings as well as adjacent to them. In

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**Figure 1-46.** 3\textsuperscript{rd} principle stress change [MPa] along a cross-section through the East ore body (looking NE); (a) @ 60 yrs, (b) @ present day. Positive values are increased tension; negative values are increases in compression.

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**Figure 1-47.** 3\textsuperscript{rd} principle stress change [MPa] at present day; (a) Elevation = 1000 [m], (b) Elevation = -400 [m]. Positive values are increases in tension; negative values are increases in compression.
the lower half of the mine there are decreases in 3rd principle stress but only in small regions adjacent to the East and West ore bodies and 13&15 Ledges. Typically increases in 3rd principle stress are greater than decreases in stress.

The largest increases in 3rd principle stress occur in the ore bodies in the deepest regions of the East and Main ore bodies and in the 13&15 Ledges. In these regions the increase in stress can exceed 30 MPa. Near the ground surface the 3rd principle stress in the ore regions increases by 1 to 9 MPa and is highly variable (Fig. 1-46 & 1-47).

Increases in stress above the workings primarily occur in the upper regions around the mine at depths less than approximately 700 m below ground surface (elevations > ~870 m) (Fig. 1-46). At these depths the increase in stress is typically on the order of 1 to 3 MPa. At greater depths stresses adjacent to the 5 major ore formations increase by up to 8 MPa with the greatest increases around the SE tips of the East and Main ore bodies (Fig. 1-47).

Decreased 3rd principle stress adjacent to the workings primarily occurs in the upper half of the mine. In these upper regions the decrease in stress can reach almost -9 MPa and is typically on the order of -4 to -6 MPa. The 3rd principle stress also decreases in some small regions in the lower half of the mine mainly in the region between the East and West ore bodies where the 13&15 Ledges are. In these regions stresses can decrease by almost -9 MPa. Decreased 3rd principle stress also occurs below the ore bodies and ranges from -1 to -2 MPa near the boundaries of the workings.
Change in Stress Profiles and Gradients

The principle stresses change systematically along vertical profiles through the mined-out regions. The stresses are slightly more compressive than ambient (more negative) above and below, and significantly less compressive (less negative) than ambient within the mined out regions (Fig. 1-48). The maximum changes from ambient occur at the interface between mined-out and intact rock, although this is likely an artifact of the abrupt change in material properties used in the model at this location. The perturbation in horizontal stresses extends upward to the ground surface, and downward by 500 m in the example profile shown in (Fig. 1-48). The increase in vertical compression is much smaller than the increase in horizontal compressive stresses below

Figure 1-48. Principle stress profiles with depth before mining (black line) and after mining (red line) through the East ore body. At point ‘D’, (Fig. 1-43, 1-45, & 1-47).
Figure 1-49. Principle stress profiles along cut lines B-B’ and C-C’ from (Fig. 1-43, 1-45 & 1-47), before mining (black line) and after mining (red line) through the East and West ore bodies.
the mine, and the vertical stresses above the mined-out region is essentially unaffected by the underlying workings (Fig. 1-48).

The stresses along horizontal lines generally resemble those along vertical lines through the mined-out regions (Fig. 1-49). For example, the stresses adjacent to the mined-out regions are more compressive (more negative), whereas in the mined-out region they are less compressive (less negative) than the ambient stresses (Fig. 1-49). The largest change along horizontal lines occurs in the 1\textsuperscript{st} principle stress, whereas the largest change along the vertical lines was in the 3\textsuperscript{rd} principle stress (Fig. 1-48 & 1-49). Particularly large increases, up to -40 MPa, in compression occur in the solid rock between the mined-out regions (Fig. 1-49)

Superimposed upon the horizontal stress profiles is the SW to NE stress gradient which is independent of mining. The gradient for the 2\textsuperscript{nd} and 3\textsuperscript{rd} principle stresses along a horizontal line ranges from 0.6 to 1.0 MPa/km with decreasing compression from the SW to the NE. The gradient in the 1\textsuperscript{st} principle stress is roughly the same (0.6 to 1.0 MPa/km) along the horizontal lines but with increasing compression from the SW to the NE.
**Soft Inclusions**

The stress results found with the numerical analysis are analogous to the behavior found when analyzing an idealized soft inclusion that is less stiff than the enveloping rock around it. To show this an idealized 2D stress analysis was developed that had a 10x10 m soft inclusion within a 100x100 m rock formation. The Young’s modulus for the enveloping rock was taken as the average modulus from Table 1.1 and the density of the rock and Poisson’s ratio were assumed to be the calibrated values from the Homestake analysis, 2900 kg/m$^3$ and 0.32 respectively. The modulus in the soft inclusion was assumed to be 30% of the modulus of the enveloping rock. The mechanical boundary conditions included rollers along the sides and bottom and a free surface at the top of the model. The region was loaded with a gravity load equal to the unit weight of the rock.

![Figure 1-50. Vertical stress concentrations around the soft inclusion in the idealized analysis. Stress change is in MPa.](image_url)
The analysis showed that within, above and below the soft inclusion there were reductions in compression (positive values), and adjacent to the inclusion there were increases in compression (negative values) (Fig. 1-50). This is approximately the same as the general stress response around the mine workings in the Homestake analysis. The results from the Homestake analysis do show more variation in the stress change around the mined out regions, however, this is due to the more complicated geometry of the mine workings compared to the simpler square inclusion in the idealized model.

The vertical and horizontal stress profiles from the Homestake analysis (Fig. 1-48 & 1-49) and the idealized soft inclusion analysis (Fig. 1-51) are also approximately the same. The vertical stress depth profile from the Homestake analysis (Fig. 1-48) and the idealized analysis (Fig. 1-51) both show a very similar shape with a slight increase in stress.
(reduction in compression) directly before and after the inclusion and a larger increase in stress within the inclusion. The horizontal profiles from the two analyses also show approximately the same behavior when going through the ore regions. Each shows a sharp decrease in stress (increased compression) at the outside boundaries of the inclusion and then large increases in stress (reduced compression) at the inside boundary of the inclusion that decreases towards the center of the inclusion (Fig. 1-49 & 1-51).
Deformation Due to Mining

Figure 1-52. Vertical displacement change [m] due to mining along a cross-section through the East ore body (looking NE) at various times: (a) 20 yrs, (b) 40 yrs, (c) 60 yrs, (d) 100 yrs, (e) 125 yrs, (f) 135 yrs.

Mining activities resulted in notable deformation above and adjacent to the mine workings. The largest deformation that occurs because of mining is vertical subsidence above the mine workings, Fig. (1-52 & 1-53). This deformation results from softening of the workings region and reduction in the hydraulic pressure in the workings. Subsidence
begins within the region representing the open pit and upper workings of the mine (Fig. 1-52). As mining progresses the region of subsidence expands laterally out from the mine and downward with the largest amount of expansion in the SE direction, towards the lower portions of the workings, Fig. (1-52 & 1-53). The greatest subsidence occurs at the surface within the upper workings. At depth in the mine the greatest subsidence occurs along the hanging wall of the mine workings.

At the ground surface, the magnitude of deformation when the mine is fully dewatered ranges from -0.18 m in the region of the upper workings to less than -0.01 m at several kilometers away. This region is approximately bounded by an ellipse with an apex about the center of the upper workings and a long axis approximately S30E (Fig. 1-53). The short axis of the ellipse is approximately 2.7 km, and the longest limb is oriented S30E and is approximately 5.3 km. With depth the shape and orientation of this region stays fairly consistent but there is a slight increase in the extent of the horizontal region. The maximum subsidence around the mine in general decreases with depth (Fig. 1-52 & 1-53). Near the center depth of the mine (elevation ~ 370 m) the maximum subsidence is approximately -0.13 m (Fig. 1-53). Maximum subsidence at the base of the mine is approximately -0.037 m (Fig. 1-53).

Mining activities also result in vertical uplift. Uplift typically occurs along the footwall of the mine workings, with the largest uplifts occurring in the lowest regions. The uplift along the footwall increases from a few tenths of a mm at shallow depths to approximately 10 mm at the bottom of the mine (Fig. 1-52 & 1-53). The maximum uplift at the end of mining and dewatering is 13 mm in the West Ore Body. Additional uplift
Figure 1-53. Plan view maps of the vertical displacements, in meters, due to mining at various elevations in the mine when the mine is fully dewatered; (a) 1565 m, (b) 970 m, (c) 370 m, (d) -230 m, (e) -800 m.
occurs when the pressure increases during filling of the mine. A maximum uplift of 22 mm occurs when the water reaches its highest level, according to the simulations (Fig. 1-52 & 1-53).

**Horizontal Displacements due to Mining**

Along with vertical subsidence and uplift there are also displacements in the horizontal direction due to mining activities. Horizontal deformation is similar in magnitude to vertical uplift but is considerable less than vertical subsidence magnitudes (Fig. 1-54 & 1-55). Horizontal displacements are typically a few cm or less with a maximum displacement of over 5 cm. Typically the greatest displacements are near the

![Figure 1-54. Plan view maps of the East-West displacements [m] due to mining at several depths; (a) 1565 m, (b) 370 m, (c) -800 m. Positive values represent displacement to the East and negative values represent displacement to the West.](image)
ground surface and decrease with depth. Horizontal displacements around the mine occur inward towards the workings.

The region around the mine workings where horizontal displacements of at least 1 mm occur at the ground surface extends out from the workings by over 5 km at its maximum to the East and South (Fig. 1-54 & 1-55), and the extent of this region shrinks with depth. Near the base of the mine this region extends out at its maximum by roughly 3 km to the NW and SE (Fig. 1-54 & 1-55). Horizontal displacements occur in a region that is more variable than the elliptical region where vertical displacements occur. The

Figure 1-55. Plan view maps of the North-South displacements [m] at several depths (a) 1565 m, (b) 370 m, (c) -800 m. Positive values represent displacement to the North and negative values represent displacement to the South.
areas where horizontal displacements occur tend to be non-uniform and are largely controlled by the geometry of the workings (Fig. 1-54 & 1-55).

Maximum displacements in the E-W directions typically exceed those in the N-S directions. In the upper portions of the mine the maximum E-W displacements are greater than the maximum N-S displacements by up to several cm (Fig. 1-54 & 1-55). In the mid-elevations of the mine the E-W displacements typically are greater than the N-S displacements by up to 1 cm or less (Fig. 1-54 & 1-55). In the lowest elevations of the mine the E-W displacements exceed the N-S displacement by only several mm (Fig. 1-54 & 1-55).

The maximum E-W displacement near the ground surface ranges from 5.4 to -3 cm (Fig. 1-aa). Positive values for the E-W displacement represent displacement to the East and negative values represent displacements to the West. Maximum E-W displacements at the base of the mine range from 2 to -2.2 cm (Fig. 1-54). The maximum N-S displacements near the ground surface range from 3 to -3.3 cm (Fig. 1-55). Positive values for the N-S displacement represent displacements to the North, and negative values represent displacements to the South. Maximum N-S displacements near the base of the mine range from 1.3 to -2.1 cm (Fig. 1-55).
Hydraulic Head, Fluid Flow, and Stress along the Fault

The Homestake fault is located to the SW of the major ore regions and intersects the West Ore Body. The diameter of the fault is approximately 1500 meters, with a strike of N20W and a dip of 60° NE, and is centered at (152, -2134, 55) m (Fig. 1-56). The fault is modeled as a planar feature that is approximately two orders of magnitude more conductive than the surrounding rock with an effective aperture of $1 \times 10^{-4}$ m. The effective aperture of the surrounding rock around the fault is approximately $1 \times 10^{-5}$ m. It is assumed that the fault will behave as a fast track for fluid flow to the mine and will also have changes in stress along its surface because of mining.

Mining activities result in changes along the Homestake fault, including changes to the hydraulic head as well as the normal and shear stresses along the fault. Initially the hydraulic head along the fault is the same as the head in the surrounding rock. Before mining the hydraulic head along the fault varies from roughly 1600 m to 1620 m, with the highest head along the southern edge of the fault and lowest head along the northern edge (Fig. 1-58). Hydraulic head along the Homestake fault decreases as a result of dewatering the mine (Fig. 1-57 & 1-59).
As the upper workings are dewatered the hydraulic head along the northern edge of the fault begins to decrease first (Fig. 1-59). In the first 20 years the maximum head reduction along the fault is roughly -5 m and by 40 years the head along the fault has been reduced by -50 m (Fig. 1-59). As mining progresses and the mine bottom reaches depths close to the elevations along the fault plane, around 60 years, head along the fault is more greatly affected (Fig. 1-57 & 1-59). After 60 years of mining the head is reduced by -350 m along the northern edge of the fault (Fig. 1-59b). The head along the fault has larger reductions, -800 m, when the depth of the mine reaches the level were the fault first intersects the West Ore Body, at roughly 68 years and a depth of 1500 m below ground surface (Fig. 1-59c). Once the mine depth reaches the intersection between the fault and West Ore Body the greatest reduction in
hydraulic head occurs within the intersection (Fig. 1-59c-f). When mining operations reach the elevations where the fault and West Ore Body intersect the hydraulic head along the fault at the intersection is approximately equal to the head in the West Ore Body. When the mine reaches the total depth of 2.4 km, at approximately 100 years, the greatest reduction in hydraulic head along the fault is approximately -2000 m.

Figure 1-59. Hydraulic head change [m] along the fault plane at various times in the simulation; a) 20 yrs, b) 60 yrs, c) 68 yrs, d) 100 yrs, e) 125 yrs, f) 135 yrs.
The hydraulic head along the fault increases when the mine begins refilling. At the height of refilling the change in head along the fault compared to pre-mining conditions is approximately -1400 m. When the mine is dewatered again the largest reduction in head along the fault returns to approximately -2000 m.

The pattern of the head change along the fault is controlled by the faults’ location to the major ore bodies and the depth of the lowest hydraulic head in the workings. The greatest reduction in head along the fault before the depth of the mine reaches the intersection between the fault and West Ore Body occurs along the fault edge nearest to the region of lowest hydraulic head in the workings. After the mine has reached the depth of the intersection, the greatest reduction in head along the fault occurs in the intersection. The reduction in hydraulic head along the fault early on, when dewatering has only occurred in the upper portions of the East and Main ore bodies, is greatest along the northern edge of the fault towards these regions (Fig. 1-59a). As mining progresses and the region of lowest hydraulic head in the workings moves downward and to the SE, the region of greatest reduced head along the fault rotates to the East, still occurring along the fault edge (Fig. 1-59b). When the depth of the mine ranges from the shallowest to the deepest contacts of the fault and West Ore Body intersection, the largest reduction in head occurs in the intersection at the elevation of the lowest head in the West Ore Body (Fig. 1-59c). The largest reduction in head along the fault occurs along the deepest contact of the intersection (E-SE edge of fault) once the region of lowest head in the mine falls below the elevation of the contact (Fig. 1-59d). Once the region of lowest head in
the workings is below this contact, a fairly stable pattern of head change along the fault develops with changes primarily to hydraulic head magnitude occurring (Fig. 1-59d-f).

**Fluid Flow along the Fault**

Water from the vicinity of the mine region is captured within the fault plane. This was found using traces of fluid particles that were positioned around the mine by up to 4 km away and at elevations that range from the surface of the model (1570 m) to ~800 m.

Particle traces show that water captured by the fault typically originates from regions to the SW of the fault and West Ore Body at distances of 2-3 km away and between elevations of roughly 1200 m to 0 m. These traces cannot be analyzed easily using still capture figures and require a dynamic figure that can be rotated to fully show this behavior and therefore they were not included in this document.

In general fluid captured along the fault plane flows along the fault towards the West Ore Body where it discharges to the mine workings (Fig. 1-60). Fluid discharging to the West Ore Body that is captured by the fault typically has travel times on the order of 50 to 80 years. Fluid that has originated from approximately the same region but is not captured by the fault before discharging to the West Ore Body typically has travel times...
that are greater than this by up to 50 years. Generally, fluid discharging to the West Ore Body that flows down the fault plane has longer travel paths than fluid that does not flow down the fault plane. This suggests that the three order of magnitude increase in transmissivity along the fault when compared to the host rock, caused by the larger fracture aperture, causes the fault plane to behave as a fast path for fluid to the workings with higher flow velocities.

**Stress along the Fault**

Stresses along the fault before mining are controlled by rock properties and depth. Mechanical rock properties along the fault are assumed to be the same as those of the host rock. The stresses along the fault before mining are the same as those in the surrounding rock. The two primary stresses along the fault that are of interest are the normal stress, stress perpendicular to the fault plane, and shear stress, stress parallel to the fault plane. Increasing values of normal stress represent reductions in compression perpendicular to the fault plane and decreasing values of normal stress represent increases in compression perpendicular to the fault plane. The shear stress in the analysis is presented as the total shear stress and the individual components are not considered, therefore the shear stress values are always positive.
**Normal Stress along the Fault**

The normal stress along the fault decreases (becomes more compressive) with depth before mining occurs (Fig. 1-61a). At the highest elevations along the fault plane the normal stress is approximately -22.3 MPa and reaches approximately -40.5 MPa at the deepest elevations (Fig. 1-61a & 1-62). After mining the normal stress is approximately -23.2 MPa along the shallowest edge of the fault and -41.4 MPa along the deepest edge (Fig. 1-61b & 1-62). After mining the normal stress within the intersection of the fault and West Ore Body ranges from approximately -24 to -28 MPa (Fig. 1-61b).

Mining activities cause increases and reductions in the normal stress on the fault plane. The normal stress along the fault increases (becomes less compressive) on the shallowest edge of the fault (W-SW edge) and decreases (becomes more compressive) on the N-NW edge of the fault during the first few decades of mining (Fig. 1-63a). The maximum increase in normal stress along the fault after 20 years is roughly 0.04 MPa, and the maximum decrease in normal stress is approximately -0.075 MPa (Fig. 1-63a).

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Figure 1-61. The normal stress [MPa] along the fault before mining began (a) and after mining (b). East is out of the page and the fault dips to the NE.
Between approximately 40 and 60 years after mining begins, the normal stress along the fault decreases along the entire fault surface. The maximum decrease in normal stress along the fault at approximately 60 years reaches -1.25 MPa along the northern edge of the fault (Fig. 1-63b). The normal stress along the contact between the fault and the West Ore Body increases when the mine reaches the depth of the intersection. At 68 years when the depth of the mine first reaches the contact between the two domains, the normal stress increases along the intersection by approximately 6.2 MPa (Fig. 1-63c). The region of increased normal stress in the intersection expands as mining progresses (Fig. 1-63c-d). When mining has progressed past the deepest contact between the fault and West Ore Body, the entire intersection has an increase in normal stress that can reach approximately 10.5 MPa by the end of mining along the northeast edge of the contact (Fig. 1-63f).
Figure 1-63. The change in normal stress along the fault at (a) 20 yrs, (b) 60 yrs, (c) 68 yrs, (d) 75 yrs, (e) 100 yrs, and (f) 125 yrs. East is out of the page and the fault dips to the NE.
The normal stress outside of the contact between the two domains has a reduction in normal stress compared to pre-mining conditions that can reach -4.5 MPa. This maximum decrease in normal stress occurs directly adjacent to the shallowest portions of the contact between the fault and West Ore Body. Typically, the decrease in normal stress outside of the contact is approximately -2 MPa.

The pattern of normal stress change is most variable before the depth of the mine reaches the contact between the fault and West Ore Body. In these earlier times, before 68 years, the greatest changes occur typically along the NW edge of the fault with reduced change outward along the fault from this edge (Fig. 1-63a,b). Directly before the depth of the mine reaches the contact the shape of the affected region is roughly parabolic (Fig. 1-63b). After the depth of the mine has moved below the fault plane and contact, the pattern of stress change along the fault becomes more uniform with increased stress in the contact and decreased stress outside of this region (Fig. 1-63e,f).
Shear Stress along the Fault

Shear stresses along the fault are considerably less than the normal stresses both before and after mining (Fig. 1-64a). The shear stress along the fault ranges from 7.3 MPa along the shallowest edge of the fault to 18 MPa along the deepest edge before mining (Fig. 1-64a & 1-65). After mining the shear stresses range from 5 MPa in the intersection of the fault and West Ore Body to 22.3 MPa along the fault’s deepest edge (Fig. 1-64b & 1-65). Along the shallowest edge of the fault the shear stress is approximately 8 MPa (Fig. 1-64b).

Figure 1-64. The shear stress [MPa] along the fault before mining began (a) and after mining (b). East is out of the page and the fault dips to the NE.
Mining results in both increases and decreases to the shear stress along the fault. In the first 40 years of mining the pattern of shear stress change along the fault includes increased shear along the northern edge of the fault and reduced shear along the SW edge of the fault (Fig. 1-66a). At 40 years the maximum increase in shear is approximately 0.17 MPa and the maximum decrease is approximately -0.05 MPa (Fig. 1-66a). After approximately 50 years this pattern changes, with increasing shear along the entire fault. Increased shear along the fault by 60 years ranges from 0.07 to 0.58 MPa with the greatest increases along the N-NW edge of the fault (Fig. 1-66b).

When the mine reaches the depth of the contact between the fault and West Ore Body, at approximately 68 years, there is a large increase in shear adjacent to the W-SW boundary of the intersection and a large decrease in shear within the intersection (Fig. 1-66c). The maximum increase in shear at this time is approximately 3.5 MPa, and the maximum decrease is -5.8 MPa. As the depth of the mine progresses through the contact of the fault and West Ore Body the shear stress in the intersection and along and adjacent to the NW boundary of the intersection decreases (Fig. 1-66d). During this same period,
Figure 1-66. The change in shear stress along the fault at (a) 40 yrs, (b) 60 yrs, (c) 68 yrs, (d) 75 yrs, (e) 100 yrs, and (f) 135 yrs. East is out of the page and the fault dips to the NE. Pt. C location is given by black x-mark.
there is increased shear adjacent to the SW and NE boundaries of the contact with the greatest increases directly along the boundary, reaching almost 7 MPa (Fig. 1-66d).

After the mine reaches its total depth, the pattern of shear stress change along the fault stays fairly consistent. There is reduced shear in the intersection and adjacent to the NW boundary of the intersection and increased shear along the rest of the fault (Fig. 1-66e,f). Within the intersection the reduction in shear stress can exceed -11 MPa. The greatest increase in shear occurs along the SW and NE boundaries of the contact between the fault and West Ore Body and can exceed 5 MPa. Along the majority of the fault the shear stress is increased by 1 to 2 MPa.
DISCUSSION

The current numerical simulation of the Homestake mine supports many of the findings from the previous model by Murdoch et al. (2012) as well as gives new insights that could not be obtained using the previous model.

The new analysis confirms that there are four major hydrologic zones in the vicinity of the Homestake mine. These include a Shallow Flow System in the upper few hundred meters of the model, a Recharge Capture Zone, a Storage Capture Zone at greater depths, and the mine workings themselves. The new analysis also shows that there is flow from the shallow to deep flow systems. The current model confirms that the deep flow system removes a small fraction of the ambient recharge to the shallow flow system and the classic cone of depression where the water table drops, pores are drained and an extensive vadose zone is formed does not occur around the mine. The previous model by Murdoch et al. (2012) also suggested that mining and dewatering resulted in a roughly elliptical region of subsidence around the mine which is consistent with the current findings.

Improvements in the current analysis allow for new insights about the hydrology and stress state in the vicinity of the Homestake mine. Improved spatial resolution and inclusion of anisotropic mechanical parameters allow the current model to be calibrated using in-situ stress measurements, which enables an estimate of the stress field in the vicinity of the mine. This gives insight into the ambient stress field, as well as how stresses are changed and concentrated during mining. Furthermore, improving understanding of the stress state improves the ability to estimate strain/deformation in the
vicinity of the mine. The improved geometry (mine workings and stream traces) refines understanding of the hydraulic head distribution, flow paths, and fluid travel times in the vicinity of the mine. Inclusion of the streams also shows the regional hydraulic head gradient in the modeled region, which was not discernible in the previous model. With the inclusion of the Homestake fault, the new model is capable of estimating the effects the fault has on the hydrology around the mine. Inclusion of the fault also makes it possible to estimate the fluid pressure and stress distribution along the fault and changes to the critical stress along the fault plane caused by mining.

**The Shallow Flow System**

Hydraulic heads in the Shallow Flow System are affected by mining activities in regions above and adjacent to the open pit region and upper workings, and in areas above the SE portions of the workings (Fig. 1-29). Outside of these regions there is little change in the hydraulic heads compared to conditions before mining occurred. The majority of water that enters the Shallow Flow System as recharge discharges to the streams; however, a small portion of this water enters the underlying Recharge Capture Zone and discharges to the mine workings. Travel times in the Shallow Flow System are typically less than 30 years.

To find the flux of water from the Shallow Flow System to the Recharge Capture Zone the vertical hydraulic flux was calculated using Eq. (1.30).

\[
q = K \frac{\partial h}{\partial z}
\]  

(1.30)
where $q$ is the hydraulic flux, $K$ is the hydraulic conductivity, and $dh/dz$ is the vertical hydraulic gradient. The vertical flux above the majority of the mine workings typically ranges from $1 \times 10^{-9}$ m/s to $4 \times 10^{-9}$ m/s. However in regions in and directly above the upper workings this flux can be higher, for example above the upper portions of the Main Ore Body the vertical flux can reach almost $3.5 \times 10^{-8}$ m/s. To estimate what percentage of baseflow, $5 \times 10^{-9}$ m/s as calculated from the hydrograph analysis, discharges to the streams and what percentage discharges to the mine the flux ratio ($q_{\text{mine}}/q_{\text{streams}}$) was found. The analysis suggests that above the majority of the mine the flux ratio ranges from 0.2 to 0.8, and therefore in these regions the mine receives 20 to 80% of the total recharge to the system. Where Whitewood creek passes over the workings the flux ratio is roughly 0.4 or less, which suggests the stream receives over 50% of the recharge. These findings imply that the rate of groundwater discharge to the creeks above the workings, including Whitetail, Whitewood, and Yellow Creeks, is diminished by discharge to the mine, but the effect from mining is insufficient to cause them to lose water. This is consistent with observations in the region which suggest that mining has not resulted in observable changes in flow in the streams overlying the mine workings (Rahn and Roggenthen 2002). In the regions above the upper workings where the downward flux exceeds baseflow, the mine receives all of the recharge to the system. Another interesting finding from the analysis is that fluxes to the mine are greater in upland regions, where streams are not present, than below the streams.

These results confirm the findings of Murdoch et al. (2012), who estimated the maximum flux ratio to be 0.8 and the flux ratio beneath the Whitewood Creek to be 0.6.
The Murdoch et al. (2012) model did not explicitly include streams, so the new model indicates that this level of detail has limited effect on the calculations of how flux is distributed between the Shallow Flow System and the mine.

**Flow Systems at Depth**

Water is captured by the mine from the Recharge and Storage Capture Zones. The Recharge Capture Zone consists of water captured by the mine that has entered the subsurface as recharge since mining began. The Storage Capture Zone consists of water captured by the mine that comes from storage in the host rock.

The capture zones around the mine were designated by plotting fluid particle traces along NW-SE and NE-SW cross-sections through the workings and determining to where the particles discharged. The regions containing particles that were captured by the mine during the 135 years of the simulation were considered to be within the capture zones of the mine. Regions where the particles did not make it to the mine in the 135 years or that discharged to the stream systems were considered to be outside of the capture zones. The Storage Capture Zone was designated as the region where particles that had been released at t=0 years discharged to the mine in the 135 years. The Recharge Capture Zone was designated as the region where particles from in or at the base of the Shallow Flow System, which were released after mining began, discharged to the mine.

Both of these zones create an elliptical region when projected to the surface. The Recharge Capture Zone creates an ellipse that is approximately 6 x 3.6 km with a major axis in the NW-SE direction (Fig. 1-67). To the NW of the open pit the Recharge
Capture Zone extends downward by roughly 1 km and then curves horizontally to discharge laterally to the workings (Fig. 1-38). To the SE of the open pit the Recharge Capture Zone extends downward approximately 2 km where it intersects the underlying workings at the SE most tip of the East and Main Ore Bodies (Fig. 1-38). To the SW and NE of the workings the downward extent of the Recharge Capture Zone varies from approximately 1 to 2 km along the main axis of the mine. In three dimensions the Recharge Capture Zone resembles an ellipsoid rotated about a SW-NE axis approximately parallel to the main axis of the mine and truncated at the base of the Shallow Flow System. The Storage Capture Zone creates an ellipse that is approximately 8.3 km x 6.6 km with a centroid that is skewed approximately 1 km to the SE of the centroid of the ellipse created by the Recharge Capture Zone (Fig. 1-67). The Storage Capture Zone extends downward from the surface by approximately 5 km (Fig. 1-38). In three dimensions the Storage Capture Zone resembles a sphere truncated at the base of the Shallow Flow System.
The surface projections of the centroids of the Recharge and Storage Capture Zones are nearly 1 km SW of the approximate centroid of the mined out regions. (Fig. 1-67). This is because the regional flow gradient from the SW to NE interacts with the flow induced by mining. This results in the mine capturing water from a larger region to the SW than the NE.

The size and shape of the Recharge and Storage Capture Zones change with time. Initially when dewatering begins water discharging to the mine comes primarily from storage adjacent to the mine. As mining progresses the Recharge Capture Zone grows downward and spreads laterally out from the mine, and the rate at which water is removed from storage diminishes. The Storage and Recharge Capture Zones continue to grow until the end of the study period. Water outside of the capture zones by the end of the study period has been affected by mining, but has not yet been captured by the mine and therefore these regions are not included in the capture zones.

The largest fluxes from the Shallow Flow System to the Recharge Capture Zone come from areas adjacent to the open pit and above the mine. The maximum fluxes occur in and above the shallowest portions of the workings and can exceed $3.5 \times 10^{-8} \text{ m/s}$ in the vicinity of the shallow workings. Typically, recharge flux above the workings ranges from $1 \times 10^{-9} \text{ m/s}$ to $4 \times 10^{-9} \text{ m/s}$. In regions adjacent to the areas directly above the workings recharge fluxes to the Recharge Capture Zone are generally $1 \times 10^{-9} \text{ m/s}$ or less especially in areas near streams. The fluxes in the vicinity of groundwater divides tend to be higher than near the streams and in areas and can reach almost $2 \times 10^{-9} \text{ m/s}$. 
One particular case is the region to the SE of the mine where the tailings pond is located. In this area recharge flux is approximately $2 \times 10^{-9}$ m/s. The simulation shows that water in the tailings pond does enter the Recharge Capture Zone and discharges to mine workings (Fig. 1-39).

From this simulation it can be inferred that the majority of recharge flux that discharges to the mine comes from three primary places, areas adjacent to the open pit region, from areas directly above the workings, and from areas in the vicinity of the mine notably near groundwater flow divides. Furthermore, these findings suggest that, at least initially, the largest amount of recharge flux to the mine comes from above and in the vicinity of the shallower portions of the workings. It seems likely, however, that if the Recharge Capture Zone continues to grow as dewatering continues, the percentage of recharge flux from areas in the vicinity of the mine, especially near groundwater divides, will increase.
Cone of Depression

Studies by Rahn and Roggenthen (2002) and Zhan and Duex (2010) hypothesized that dewatering the Homestake mine would create a large cone of depression with a radius of roughly 2 km and a maximum drawdown of approximately 2.4 km. The two studies assumed the cone of depression would resemble a truncated cone with a bottom at the lowest elevations of the mine that sloped at 41° to 45°, similar to that shown in the schematic in (Fig. 1-68). The numerical analysis shows that a cone of depression where the water table drops, (Fig. 1-68), is unnecessary to explain the available data. Instead, dewatering the mine reduces the pore pressure, but it remains above atmospheric pressure in most locations. As a result, the water table (defined as the surface where $p=0$) is only depressed by 10s to 100 m (Fig. 1-69). The simulation shows that the mine behaves like a well beneath a constant head boundary similar to a well beneath a lake or stream. In
this case, the hydraulic head at the bottom of the shallow flow system is approximately constant. Murdoch et al. (2012) assumed \textit{a priori} that a constant-head boundary could be used to represent the shallow flow system, and this analysis confirms the validity of that assumption. The lateral distance away from the mine where heads have been affected is approximately 2 km, which is consistent with the scale described by Murdoch et al. (2012).

\textbf{The Homestake Fault}

Dewatering the mine reduces the fluid pressure along the Homestake fault by -5 MPa to -20 MPa. The change in fluid pressure and the change in mechanical parameters in the West Ore Body causes the normal stress along the fault plane to change. The greatest reduction in pore pressure occurs in the mined out region of the West Ore Body (Figure 1-70). Along the majority of the fault the normal decreases (becomes more compressive) by approximately -2 MPa along the majority of the fault. The change in normal stress depends on location, however, and it increases (becomes less compressive) by approximately 9 MPa in the vicinity of the West Ore Body. The shear stress increases by 1 or 2 MPa across most of the fault and by as much as 5 MPa adjacent to the mined out region of the West Ore Body. Within the mined out region the shear stress is reduced by...
as much as -11 MPa. It is assumed that the fault is more permeable than the host rock by two orders of magnitude and locked by friction.

Reactivation of the Homestake Fault as a sudden slip could cause risks to the facility, but it also has the potential as an important scientific opportunity. An experiment has been proposed where a slip would be nucleated on an existing fault to advance the understanding of earthquakes (Germanovich et al. 2011). The Homestake Fault is a candidate for this experiment, so the conditions required for shear failure were evaluated. An estimate of the shear stress required to cause failure of a frictional material is given by

$$\tau_f = \mu_s (\sigma_n - P_f) + C_c$$

where $\mu_s$ is the static frictional coefficient, $\sigma_n$ is the normal stress, $P_f$ is the fluid pressure, and $C_c$ is the cohesion or cementation strength of the fault (Byerlee 1978, Sibson 1994, Barton et al. 1995). To estimate the critical shear stress along the Homestake fault the cohesion was assumed to be negligible, $C_c = 0$, and the frictional coefficient was assumed to be $\mu_s = 0.85$. These values are consistent with the findings from Byerlee (1978) for faults with normal stresses less than 200 MPa.

Before mining the fluid pressure along the fault ranges from roughly 9 to 21 MPa, the total normal stress varies from -22 MPa to -40 MPa (Fig. 1-61a), and the shear stress ranges from 7 to 18 MPa (Fig. 1-64a). These variables change with depth. The critical shear stress, $\tau_c$, ranges from approximately 11 to 16 MPa, according to Eq. 1.40 (Fig. 1-72a). The potential for failure was characterized using the ratio of shear stress to $\tau_c (\tau/\tau_c)$. Values of this ratio that are equal to or greater than 1 suggest that the shear stress along
the fault exceeds the failure stress and slip is likely. Values less than 1 suggest that the fault is stable (Fig. 1-72a’,b’). This ratio ranges from 0.66 to 1.1 before mining with an increasing ratio with depth. Findings from the analysis suggest that before mining the lower portions of the mine could have been near failure. To determine the potential direction of slip along the fault before mining the surface traction vector was plotted along the fault plane (Fig. 1-71).

The surface traction force is the internal force per area along the fault, and the traction vector gives the direction the force is directed. The majority of the force is into the fault plane because of the high normal compression; however, there is a component directed down along the fault plane from high elevation to low. This implies that the fault would most likely slip downward if failure occurred. This further suggests that the fault is most likely a normal fault where the hanging wall has slipped downward from the footwall.

Figure 1-71. The surface traction vector (black arrows) along the fault plane. Looking normal to the fault plane.
After mining the fluid pressure drops, and the normal stress decreases (becomes more compressive) along most of the fault and increases (becomes less compressive) in the vicinity of the West Ore Body. The shear stress increases along most of the fault and decreases in the vicinity of the West Ore Body. The fluid pressure ranges from approximately 1 to 7 MPa, the normal stress ranges from -23 to -41 MPa (Fig. 1-61b), and the shear stress ranges from 5 to 22.3 MPa (Fig. 1-61b). The critical shear stress after mining from Eq. 1.40 ranges from 16 to 33 MPa, and the ratio $\tau/\tau_f$ along the fault ranges from approximately 0.2 to 0.7 (Figure 1-72 b,b').

Figure 1-72. The critical shear stress (Panels (a) and (b)) and the ratio of the shear stress and $\tau_f$ (Panels (a') and (b')) along the Homestake fault plane before and after mining.
Mining reduces the potential for slip along the fault, according to the simulations. This is primarily due to the reduction in fluid pressure and increase in the total normal stress compression which causes the critical shear stress to increase (Fig. 1-72b). The greatest decrease in failure ratio occurs in the vicinity of the West Ore Body, which is a result from the large reduction in shear stress that occurs there. The reduction in shear stress in the vicinity of the West Ore Body is a result of the changing mechanical parameters in the ore body which causes stress change along the fault where it intersects this region.

**Deformation**

Deformation is caused by two primary processes, softening of the ore regions and changes in fluid pressure during dewatering and refilling. Softening of the ore bodies and reductions in fluid pressure both cause the mined-out regions to contract, and this controls the displacement and stresses in the enveloping rock during the majority of the analysis. Subsidence and uplift can both result from contraction of the ore regions. Subsidence is greatest in the shallowest regions of the workings and decreases with depth because the maximum subsidence at the surface results from accumulation of downward displacement through the entire depth of the mine (Fig. 1-52 & 1-53). This same effect is responsible for the increase in uplift with increased depth. The maximum uplift at the base of the mine results from the accumulation of upward displacement through the entire depth of the mine (Fig. 1-53). During refill increase in the fluid pressure in the mine workings and surrounding rock results in expansion of the rock, causing increased uplift (Fig. 1-52).
Deformation from this analysis supports some of the findings from Murdoch et al. (2012); however, there are differences between the results of the two models. The region of subsidence envelopes an area similar to that found by Murdoch et al. (2012) and is approximately the size of the Recharge Capture Zone. The largest subsidence also occurs in the upper portions of the mine with the greatest deformation near the southern end of the shallow workings as Murdoch et al. (2012) found. However this analysis suggests that the maximum subsidence is more than 3x greater than the 0.05 m estimated by Murdoch et al. (2012). The previous model also suggested that there was uplift above the lower SE portions of the mine that was greatest, ~ 0.002 m, near the ground surface. The current model suggests that vertical uplift occurs only along and below the footwall of the mine workings and is10x the previous estimate, ~ 0.02 m. The previous study by Murdoch et al. (2012) included poroelasticity but did not include softening of the mine workings, which is one of the primary reasons for the difference in the magnitudes of displacement as well as location of uplift in the two studies.

**Stress in the Mine Region**

Stresses in the region of the mine are compressive, and compression increases with depth. The magnitude of the compressive stresses is altered because of mining and is typically reduced within, above and below the workings, but compression is increased laterally adjacent to the workings. The stopes and tunnels created during the mining process reduce the effective elastic modulus when the effects of individual cavities are averaged over several hundred meters. This causes the load that was carried by the ore-bearing region to be transferred to the adjacent rock, which results in an increase in the
vertical compression in rock adjacent to the workings (Fig. 1-43). This transfer of load to the adjacent rock also reduces compression within, above and below the workings. A similar redistribution of stresses occurs in the vicinity of open cavities (Kaiser et al. 2001, Martin et al. 2003), but the scale of the mesh used in the model is too great to consider individual cavities.

Compression adjacent to the workings is increased by up to 40 MPa, with the greatest increases occurring between the East and West ore bodies and the 13&15 Ledges near the base of the mine. In these regions compressive stress in the solid rock can reach up to 90 MPa. It would be in these regions that the greatest potential for rock failure due to compression exists, and these would be important regions when designing safety measures. The compressive strength of metamorphic rocks can be highly variable and can range from approximately 10 MPa to over 160 MPa (Johnson and Degraff 1988). Measured compressive strengths from Homestake mine range from 11 to 111 MPa with an average strength of roughly 63 MPa (Pariseau and Duan Undated). Therefore under the stress conditions estimated from the numerical simulation there is potential for compressive failure. These findings are consistent with observations of instability and rock bursts within the mine (Golder Assoc. 2006, Tesarik et al. 2002)

**Future Improvements**

The design of the current model limits some of the insights that can be obtained. The upper surface of the current model is flat, and this lack of topography affects the details of flow in the shallow flow system. It seems likely, for example, that including topography would increase the hydraulic heads at shallow depths and would reduce the
difference between the heads observed in shallow wells in the vicinity of the mine and those predicted by the model. Including topography would affect stresses in the upper portions of the model. These changes would improve the ability to predict shallow conditions in the mine.

The material properties are assumed to be anisotropic with principle axes aligned with foliation, but individual formations are omitted. The Yates Member of the Poorman Formation is more thickly bedded and may be stiffer than the other units, so including the stratigraphy could improve resolution of the model. This would require including the complicated structure in the vicinity of the Homestake mine, which has been overlooked in existing models.

The current model simulates the open pit region as a soft inclusion when in reality it is a region where the host rock has been removed. Including the open pit as a topographic feature would allow it to affect stresses in a realistic way. This could also improve how the pit interacts with surface water.

The current model takes an important step forward in resolving the mined-out regions, but the tunnels, stopes, borings, and shafts that make up the inclusions are not included explicitly. Including these features as open cavities would increase the resolution of the stresses and stress distributions throughout the mine region. This would also alter the way fluid flowed through the mine regions creating regions of open space that are completely drained (P=0) that work as fast paths for fluid surrounded by regions of rock that are partially drained with residual water still present. To do this it would require an extremely dense mesh in the mine region and because of the size of the model
this is not currently feasible because of the computational power it would require to solve.

**Conclusion**

Excavating and dewatering the Homestake mine has changed the hydrology and stress state, according to the results of a 3D poroelastic simulation that represents individual mined-out ore bodies embedded in a regional aquifer system. The simulation indicates that there is a shallow flow system typical of the Black Hills region, underlain by a deeper flow system that is caused by mining activities. There is interaction between these two systems, and groundwater discharge into the mine comes in part from recent recharge to the shallow flow system and in part from water that has been released from storage in the host rock. The deep groundwater flow system has two regions, a Recharge Capture Zone and a Storage Capture Zone. Both the Recharge and Storage Capture Zones create roughly elliptical regions that are 6 x 3.6 km and 8.3 km x 6.6 km, respectively, when projected to the surface (Fig. 1-67). Above the mine workings fluid flux from the shallow flow system to the Recharge Capture Zone typically ranges from 1x10^{-9} m/s to 4x10^{-9} m/s, but can be as great as 35x10^{-9} m/s above the shallowest workings. Downward fluxes from the shallow flow system to the underlying mine diminishes groundwater flow to the stream systems, but is insufficient to cause the streams to lose water. Dewatering the mine reduces hydraulic head around the mine region but the pore-space in the vicinity of the mine remains saturated.

Mining decreases stresses (increases compression) adjacent to the mined-out regions and increases stresses (decreases compression) within, above, and below these
workings. The magnitude of stress change in the vicinity of the mine increases with depth and near the base of the mine the vertical stresses have decreased (become more compressive) by up to -40 MPa adjacent to the mine and increased (become less compressive) by up to 30 MPa. The change in the vertical stress (1\textsuperscript{st} principle stress) is typically two times larger than the change in horizontal principle stresses (2\textsuperscript{nd} and 3\textsuperscript{rd} principle stress). The overall magnitude of the vertical stress near the base of the mine can exceed 90 MPa in regions between two or more ore bodies, which is within the measured compressive strengths of rocks in the Homestake mine. Therefore, compressive rock failure is possible in these regions.

Deformation is largely caused by reductions in fluid pressure and softening of the workings as a result of mining. Both effects cause the mined-out regions to contract, which results in both subsidence and uplift. Subsidence occurs above the mine workings, with the greatest subsidence in the vicinity of the shallowest workings. The maximum subsidence is -0.18 m at the ground surface, and it decreases with depth to roughly -0.037 m along the hanging wall at the base of the mine. There is also uplift along the footwall of the mine. A few tenths of a mm of uplift occur at shallow depths, but the uplift increases with depth to a maximum of approximately 0.013 m when the mine is fully dewatered. Uplift increases when the mine refills with water, and it is approximately 0.022 m when the level of refill water is at its highest. Deformation in the vicinity of the mine results in tilt toward the mine workings.

Mining activities have changed the stresses, and thus the potential for slip along the Homestake fault, a fault with a maximum length of greater than 1 km that occurs on
the SW side of the mine. Dewatering reduced water pressure by up to 20 MPa where the Homestake fault intersects the West Ore Body, which tends to stabilize the potential for slip. The total normal compression increases by 2MPa along most of the fault, whereas it decreases by up to 9 MPa near the West Ore Body. The shear stress increases by 1 to 2 MPa along most of the fault but it decreases by approximately 11 MPa near the West Ore body. In general, the normal compression ranges from approximately 23 MPa to 42 MPa, and the shear stress ranges from 5 to 22 MPa after mining.

The critical shear stress for failure, $\tau_f$, was calculated using Byerlee’s Law, Eq. 1.31, (Byerlee 1978), and this value was used to normalize the calculated shear stress, $\tau$, in order to estimate the potential for slip along the fault. The results of the simulation indicate that the ratio $\tau/\tau_f$ increases downward from 0.7 to 1.1 during ambient conditions. This is significant because it indicates the lower reaches of the fault were critically stressed (Townend and Zoback 2000, Zoback 2007) prior to mining. However, dewatering and softening of the mined-out regions altered the stress on the fault and decreased the ratio $\tau/\tau_f$ to 0.2 to 0.7. Thus, the simulations indicate that mining has increased stability of the Homestake fault. This is significant because an experiment has been proposed (Germanovich et al. 2011) where slip would be induced locally by altering the stresses on the Homestake fault. The results of the analysis suggest that mining has increased the stress change required to initiate slip, which will increase the effort required to induce slip. However, if slip can be induced by artificial methods, the results of this analysis indicate that propagation would likely be rapidly arrested by the current stress state.
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