HOLLISTER PROJECT

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Introduction

Great Basin Gold Ltd. (GBG) owns the Hollister Mine and operates it through the wholly owned subsidiary Rodeo Creek Gold, Inc. The property is located in northern Nevada on the Carlin Trend gold belt within the historic Ivanhoe Mining District. The mine is located on the western edge of the Northern Nevada Rift, a major north-northwest trending structural zone that has localized middle Miocene volcanic rocks and numerous epithermal gold deposits. The precious metal deposits are low-sulfidation epithermal systems characterized by banded quartz veins with electrum and silver selenides. The Hollister Mine will employ the selective underground mining method of cut and fill. This will be combined with split shooting (resuing mining method) to optimize recovery and reduce dilution. The focus of mining will be on extraction of “ounces” from the narrow veins rather than trying to maximize tons. The initial drifts will be excavated along the strike of the vein and then capped with a substantial concrete sill pillar. Back stopping with hand held drills will be used to selectively mine the vein. The stopes will be backfilled with waste rock material. With every lift, an 18 inch concrete cap will be poured onto the backfill waste rock to prevent gold values from being lost into the fill. Targeted ore production rate will be over 400 tons per day. With current ore reserves, project life is estimated to be around eight years; however, Hollister geologists view the site as one of the most prospective systems for bonanza vein deposits in Nevada. Approximately $14
million has been budgeted in 2008 for underground and surface exploration with a goal to identify additional resources, develop reserves, and extend the mine life.

District History

Mining and quarrying have occurred for thousands of years in the Ivanhoe Mining District. Prehistoric Native Americans recognized that chalcedonic silica outcroppings in this region provided a good raw material source for tool making.

Mercury prospecting and mining began in the early 1900s and continued with short breaks until 1943. Renewed interest in the district surfaced in the 1960s, and the area was actively explored for mercury, molybdenum, uranium and gold.

Between 1980 and 1992, the property was owned and/or explored by U.S. Steel Corporation, Touchstone Resources Corporation, Galactic Resources Ltd., and Newmont Mining Company.

In 1992, Newmont Exploration Ltd. (Newmont) acquired a 75% interest in the property while Cornucopia held the remaining 25%. Newmont initiated additional exploration on the property with the intent of expanding the open-pit resource, but, in 1995, Newmont concluded that the site's near-surface mineralization did not meet its size and investment criteria for further exploration or development.

In 1997, GBG acquired Newmont's 75% interest in the site; then, in 1999, GBG obtained Cornucopia's 25% interest in the site and initiated a surface exploration-drilling program to identify underground potential. Under the Newmont agreement, GBG agreed to share
Newmont's future reclamation costs for past mining at Ivanhoe which is explained below in Section 1.3.

In 2002, GBG and Hecla Ventures Corporation (Hecla) formed a joint venture project, known as the Hollister Development Block (HDB) Venture, which comprised approximately 5% of the total property within the total claim block. The HDB project included an underground exploration program undertaken by Hecla.

Since late 2004, underground exploration and test mining work has been conducted at the site, which has included drifting, crosscut work, mapping, core drilling, and collection of bulk samples for metallurgical testing and trial milling. This work involved developing approximately 5,000 feet of decline, lateral, and crosscut work, with approximately 1,000 feet of drifting along the veins.

Underground core drilling from the lateral through the veins was conducted from drill stations spaced on approximate 100-foot intervals. Over 55,000 feet of core drilling has been completed from these underground drill stations.

In 2007, GBG purchased Hecla's 50% earn-in rights in the HDB project, so again GBG has full interest in the property.

Geology

The Hollister property is located at the intersection of the Carlin Trend and the northern Nevada Rift. The Carlin Trend is a northwest-trending 80 km long metallogenic corridor; mines on the Carlin Trend have produced in excess of 933,104 kg (30 million ounces) of gold and currently delineated reserves and resources total about 2.4 million kilograms (77 million ounces). Carlin Trend type mineralization is Eocene in age. The Northern Nevada Rift is a north-northwest trending feature made up of bimodal volcanic rocks that host Miocene aged bonanza epithermal gold deposits such as Newmont’s Midas Mine, Mule Canyon, Buckskin National and Hollister. The Paleozoic stratigraphy and the Eocene magmatic pulse which is critical to the development of Carlin-type deposits to the southwest are in evidence at Hollister as are the Miocene mineralizing events.

The Miocene stratigraphic section at Hollister consists of four principal components:

- felsic volcanic flows, domes and lesser pyroclastics,
- volcanoclastic/epiclastic sediments, including re-worked tuffaceous material,
- basaltic-andesite flows,
- a lower felsic to intermediate tuffaceous sequence.

The Paleozoic stratigraphic section underlying the Tertiary rocks at Hollister is dominated by the Ordovician Valmy Formation which is part of the western siliceous eugeosynclinal assemblage and the “upper plate” of the Robert's Mountains Thrust. The formation is characterized by coarsening upward megasequences of orthoquartzites, muddy quartzites, sandy argillites and bedded to laminated argillites. Minor calcareous
siltstone and sandstone are present in some of the fine-grained facies of the Valmy Formation.

Hollister represents a low-sulphidation, epithermal gold system consisting of two distinct zones of alteration and precious metal mineralisation controlled by Tertiary versus Ordovician host stratigraphy. Bulk-tonnage, low-grade, disseminated gold mineralisation is developed in Miocene volcaniclastic and lava flow rocks, while high-grade, banded vein systems are developed in underlying competent Ordovician quartzite and argillite.

The near-surface Hollister deposit is a large low-grade gold system in Miocene volcanic rocks. This previously exploited deposit is thought to have been formed by leakage from a deeper, Valmy Formation hosted, high-grade gold feeder system. A core intercepted 0.73 m (2.4 ft) grading 1,115.6 g/t (32.53 oz/ton) gold in the west Hollister area intersected in 1994 drilling by Newmont is interpreted to be an example of a high-grade feeder vein to the overlying volcanichosted disseminated gold mineralisation. There were at least 34 such historic high-grade intersections in the Valmy Formation that required follow-up drilling to delineate the Hollister feeder vein system.

Mining

The mine plan for the Hollister Mine has been developed based on a geologic model prepared originally by Hecla and later modified by Great Basin Gold (GBG) and GeoLogix Mineral Resource Consultants (Pty.) Ltd. The general geometry of the deposit is a series of narrow veins. The goal for the mine planning effort since October 2007, was to implement the Feasibility Study’s mine design and take the property from development...
to full production including commissioning. The mine plan presented in this section represents the current base case from which future production sequencing can be examined and enhanced. It represents a snapshot as of the first Quarter, 2008.

As additional core drilling and exploration work is continued at the Hollister Mine and as geologic information and operational experience is developed and evolves, the mine management team will continue the iterative process of resource updating and mine plan optimization. A test mine was developed using an 842 m long decline at a grade of -18% to provide a better understanding of the geology and the resource, as well as to obtain bulk ore samples and to establish an exploration drilling platform. Four bulk samples have been shipped for processing. The first two bulk samples 6,074 tons containing 0.418 Au OPT, and 3,101 tons containing 0.708 Au OPT) have been processed. The remaining two samples (5,100 tons of low grade, and 5,002 tons of high grade) have been shipped and are awaiting processing. The use of a test mine has greatly reduced project and mine development risk. The current extent of the mine is shown below.
Mine Design

Overhand drift and fill were accepted as the most appropriate method for the Hollister deposit. Overhand drift and fill uses the mine waste as backfill by stacking it in the mined drift with a loader. A cemented fill cap (future floor) is poured on top of the stacked waste. The cement cap provides a good working floor and minimizes gold losses on the next cut taken from the vein above. It is intended to use two rescue variants of the method, the uppers option and the drifting option.

Overhand cut and fill with uppers will be the preferred mining method. This option incorporates drilling 1.8 m uppers in the sill drive back after adequately installing split sets and mesh for ground support. The vein is carried in the center of the sill drive. Recent excavation tests in the Clementine and Gwenivere veins demonstrate this method is far more efficient and reduces dilution over the drifting option. The initial uppers can be confined to the vein width of +/- 0.9 m, blasted and mucked before drilling and blasting the waste on either side of the 2.1 m wide drive. This waste forms the basis of the waste fill for the next lift. The waste fill is required to be filled to a height of 1.7 m.
The drifting option incorporates mining the vein on one side of the sill drive face. The wider section, that may be either the vein or the waste, is resued and mucked before slashing the narrow section. In both cases the waste must be mucked out to a re-muck bay before being re-handled back into the stope after completion of the stope for waste fill. This method will only be employed if ground control problems are induced by the uppers option.

For the drifting option, if the vein is over 1.0 m wide, but under 1.5 m wide, then the vein can be shot first and the ore removed. After the ore is removed, the waste is shot and removed. It is only when the vein gets to a width of 1.5 m that the entire vein drift width is shot at one time. This mining procedure will minimize the dilution of the resource material and minimize the cost per ounce of the gold and silver produced.

During development, the mine has identified two high grade ore shoots, one in the Clementine vein and one in the Gwenivere vein. A variant of the overhand cut and fill with uppers will be used to mine the high grade ore shoots. Small mining equipment, such as air powered slushers, will be used for mining and extraction. Only the vein will be mined and dilution will be kept to an absolute minimum. The vein will be accessed by man ways and extracted with mill hole rings. Special ore handling facilities are being designed to limit contamination and gold losses.
Ground Support

The method of installation of the required ground support will vary according to the opening size. Crews within the vein drifts will employ jackleg drills for drilling and rock bolt installation. For the ramps, level drifts and cross cuts, a jumbo will be used to place split sets and a bolter will be used to place fully grouted resin bolts.

The following table summarizes standard bolt patterns and types.

<table>
<thead>
<tr>
<th>Type Opening</th>
<th>Bolt Length</th>
<th>Bolt Type</th>
<th>Bolt Pattern</th>
<th>Other Applied Control</th>
</tr>
</thead>
<tbody>
<tr>
<td>Vein Drifts</td>
<td>4 Feet</td>
<td>Split Set-39</td>
<td>0.9 m x 0.9 m</td>
<td></td>
</tr>
<tr>
<td>Cross Cuts</td>
<td>6 Feet</td>
<td>Split Set-39</td>
<td>1.1 m x 1.1 m</td>
<td>Chain Link</td>
</tr>
<tr>
<td>Level Drifts</td>
<td>6 Feet</td>
<td>Split Set-39</td>
<td>1.1 m x 1.1 m</td>
<td>Shotcrete &amp; Galvanized wire</td>
</tr>
<tr>
<td>Ramps</td>
<td>8 Feet</td>
<td>Split Set-46</td>
<td>1.1 m x 1.1 m</td>
<td>Shotcrete &amp; Galvanized wire</td>
</tr>
</tbody>
</table>
Stope Backfilling

A reinforced concrete sill pillar is being installed on the initial cuts of the high grade portions of the veins. This will allow for eventual recovery of the crown pillars. The concrete sill pillar is 0.9 m thick and has a double row #10 rebar and wire mesh. The rebar is tied off to 1.5 m rebar bolts grouted 1.2 m into the ribs. The concrete sill pillar is poured onto an 0.5 m blast cushion constructed of loose gravel.

The backfill system for the Hollister Mine will be composed of two parts. First, waste rock will be used to fill the mined out drift by placement with a loader. Next, a cement cap will be poured on top of the placed (stacked) waste. The cement cap will provide stability for the equipment to operate on, minimize dilution by keeping the loader dipper from digging into the waste rock floor, and control loss of fine ore into the waste.

The drift to be backfilled will be prepared for filling by placing a cement slurry delivery pipeline in the back of the drift. This pipeline will have multiple spigot points along its length. The drift will be filled in 15 m sections. The mine waste will be stacked by an underground loader. The waste rock will be placed as necessary to fill the drift up to 1.7 m high. A painted marker line will be laid out by a surveyor to guide the waste rock stacking operation.

A dam or berm will be built at the end of the 15 m section and cement-sand slurry will be pumped on top of the stacked waste until a 0.6 m thickness has been obtained. The poured section will be allowed to cure, after which the dam will be removed and a further 15 m section will be prepared and poured. This will continue until the attack ramp has been reached.

Backfill will be done as an alternate cycle with mining, so that one side of the stope is mining while the other side is backfilling. Mining will continue with the next successive cut on top of the cement cap of the previous cut.
The following shows a typical vertical mining sequence.

Ore Development - Vein Drift Stope Design

The drift size for the stopes will be 2.1 m wide by 3.05 m high (7 ft. x 10 ft.) and up to 137 m (450 ft.) in strike length. The payable veins are variable and average some 90 m (300 ft.) on either side of the crosscuts. The stope lengths were controlled by the decision to divide the 850 m vein strike length into 3 sections with 3 crosscut positions after taking into account the hauling capability of the 2 yd (1.5 m³) loaders to be utilized for the ore, waste and backfill loading cycles.

Main Access System

The main access system from the surface comprises the existing exploration decline (Main Decline) mined to the 5150 level. Ramps and declines are mined 4.9 m wide by 5.2 m high (16 ft. by 17 ft.). A second decline (4705 Decline) will provide access to the remainder of the mine. This 4705 Decline branches off the Main Decline and accesses a series of levels mined at 84 m (110 ft.) vertical intervals connected by a single ramp.
system positioned on the central-eastern side of the ore zone. This development layout is shown below.

The Level Drifts in turn provide the launch platforms for the crosscuts. The vertical interval between levels was determined by a trade off exercise to optimize the waste mined in the sub levels and crosscuts using an attack ramp approach to mining. The Level Drifts are mined 4.6 m wide by 4.6 m high (15 ft. by 15 ft.). Only a small amount of this work is directly involved with stope development. Other work of this size is classed as capital development.

Crosscuts

The crosscuts are designed to access the stope cut perpendicular to the strike of the vein. Using the attack ramp method, the crosscuts are mined in sequence in ascending order starting at the lowest cut elevation and progressing upward. The initial crosscut is driven at an inclined gradient of ±16% to ensure truck hauling in 2nd gear is maintained. The cross cuts will be 4.6 m wide by 4.6 m high (15 ft. by 15 ft.) and vary in length according to the number and location of the veins being accessed.

When high grade is encountered, only the vein will be mined and dilution will be kept to an absolute minimum. The vein will be accessed by man ways and extracted with mill hole rings. Special ore handling facilities are being designed to limit contamination and gold losses. The initial crosscut will be used to access the vein. Additional drifts will driven off the crosscuts to access the raises.
Mine Dilution

The property resource is reported undiluted. The mineable resource includes dilution and recovery, which is also included in the mine plan. The primary dilution applied to the stopes from the mining method study is a constant 0.15 m (0.5 ft.) from the foot wall and 0.15 m (0.5 ft.) from the hanging wall of the vein. The dilution grade calculated from the footwall and hanging wall is zero grade gold and silver. The average ore dilution as a consequence of ingress of waste into the stope cut is 20 per cent. In narrow veins of less than 0.76 m (2.5 ft.) the dilution will be greater than 20% but in wider veins the dilution will be less. The mine dilution will vary from 28.6% for the 0.6 m wide (2 ft.) vein to 16.7% for the 1.5 m (5.0 ft.) vein. The average of 20% is based on an average vein width of 0.76 m (2.5 ft.) and assumes 0.15 m (6 in.) of waste mined on either side of the vein to give 0.3 m (1 ft.) of waste included with 0.76 m (2.5 ft.) of ore, which amounts to 20% waste dilution.

A very minor amount of ore loss will occur as a consequence of loading the rock adjacent to the friable ore vein. Gold loss to account for this loading problem is estimated to be 1.5 %. In addition, gold losses may occur between the stope and delivery to the plant. A total gold loss of 6 % has been assumed between the stope and the plant.

Mine Haulage

Load haul dump (LHDs) units of 1.5 m$^3$ (2 cubic yard) capacity will haul both the ore and the waste to separate muck bays, located close to the attack ramp entries. The ore will be picked up from the mucking bay by either the 3.0 m$^3$ (4.0 cubic yard) or 4.6 m$^3$ (6 cubic yard) front end loaders (FELs) to load 30 ton diesel trucks. The trucks transport the ore down or up the spiral ramps, across the mine on the lateral drifts and up the west decline to the ore stockpile on the surface.

The waste rock generated during mine and stope development will be hauled to surface via underground haulage trucks. During stoping operations, the waste generated by vein drifting will be stored in muck bays until used as backfill in the mined out portion of the stope. In the early stages of mining, when there is insufficient room in the muck bays, some of the stope waste rock will be hauled to the surface storage. However, most of the waste mined during vein mining operations will be used as backfill. Mining of each cut (slice of the ore) will progress from the attack ramp toward the stope limit going west (for example) along the vein. The waste mined from this activity will be stored in a waste muck bay. When mining commences on the east side of the attack ramp, the waste generated will be placed (stacked) in the west end vein drift that was previously mined on that level. Using this technique, up to 33% of the total waste generated during mining can be stored underground.
The uppers stoping method will allow the bulk of the slashed waste to provide fill for the stopes where the vein width is less than 0.9 m (3 ft.). Where the veins are wider, access development waste stored in re-muck bays will provide the make up to the 1.7 m (5.5 ft.) fill height.

The 3.0 m$^3$ (4.0 cubic yard) or 4.6 m$^3$ (6 cubic yard) diesel loaders used for development work will have push-plate dump mechanisms so that truck loading can be accomplished within a low-back environment and the waste rock backfill can be stacked effectively.

Mine Production

The average ore mining rate is 399 t (440 tons) of ore per day. While the production may seem low for those accustomed to mechanized cut and fill mining, one must remember that the resuing mining method leaves half of the stope tons mined as waste rock in the mine. In reality, the mine is producing 800 t (880 tons) per day of material from the stopes.

The ore geometry is of primary importance when determining an actual ore mining rate. To mine the three vertical veins, the amount of development that must be completed per ton of ore mined is one of the limiting factors of the sustainable mining rate for the time period that will optimize the return on the mining investment. Another limiting factor for starting up and running the mine is the recruitment and maintenance of an underground labor workforce. Being located on the far north end of the main Carlin trend, recruiting a smaller workforce producing a smaller tonnage of high grade ore will have a better chance of success than trying to recruit a larger workforce producing a larger tonnage of lower grade ore in the early years of the mine. Also, the narrow veins and fairly weak rock limit the size of equipment that can be assured of success in these stopes as demonstrated in the test mine.

Mining is to progress in successive increases up to an average rate of 400 t (440 tons) of ore per day. The initial production rate will be 30 t/d (33 tons/day), but it will reach 91 t/d (100 ton/day) by April 2008 and then increase to 400 t/d by October 2009. The mining rate will vary according to the width of the veins.

The tonnes of ore produced will also vary with the width of the vein. The average vein width is 0.76 m (2.5 ft.). It can be expected that up to 21 t/d (23 ton/d) of ore can be produced from each stope. There will be approximately 22 stopes operating at any one time during the period from 2010 through 2013.

Ore body access development and some ore drifting is currently underway. Stoping will commence in 2008 from the existing exploration lateral drift, and then from the main crosscuts developed from the exploration lateral drift. In late 2008, production commences on the next lower level. From this point on, successive sublevels are developed off the ramp at intervals of 33.5 m and 39.6 m (110 ft. and 130 ft.), depending
on whether a crown pillar of 6.1 m (20 ft.) is left. Crown pillars are left between every second level to allow for the sequencing of mining on two levels at any one time.

Mine Dewatering

The mine development from the exploration phase of the project generated 21.4 l/s (340 usgpm) of silt laden water. To effectively dispose of water encountered during mining, a Water Pollution Control Permit (Infiltration) was obtained from the Nevada Division of Environmental Protection to design, build, and operate such a water disposal system. GBG plans to install an active desilting system (sand screw, conventional thickener/clarifier and belt press) to reduce the amount of suspended sediments that report to the onsite surge ponds. The final filter system was permitted to handle up to 57 l/s (900 usgpm).

The mine at startup is expected to generate 30 l/s (474 usgpm) of water and silt that will need to be handled by the surface water system. Mine sumps to handle this amount of water and silt, have been constructed that collect the slurry. The sump pumps then send the dirty water to tanks feeding the slurry pumps. These consist of 4/3 Warman slurry pumps, pumping in three stages in series with a pipe system which lifts the slurry up the decline and to the surface catch-basin ponds.

Excess water encountered from the underground exploration workings is pumped from the mine and routed, via a buried pipeline, to Rapid Infiltration Basins, RIBs, approximately 6.5 km south of the project site. The purpose of these RIBs is to allow water pumped from the underground workings to be reintroduced to ground water.
through infiltration. Because the existing RIBs are now functioning at or near their engineered capacities, additional RIBs are needed. The new RIBs are under construction and will be incorporated into the current mine water handling system by mid-year.

Nevertheless, mining at depth will require dewatering. The mine-dewatering program will be built and operated from within the mine (in contrast to using surface wells). The underground system will consist of six, 165 mm (6½ in.) holes drilled approximately 137 m (450 ft.) deep, spaced 30.4 to 131 m (100 to 430 ft.) apart along the lateral area. The water pumped from the dewatering wells will be kept separate from the water seeping into the mine. It will be put into a pump feed tank, pumped to the surface, and discharged as surface water. This will require an NPDES permit with no additional treatment being required.

A system of three pumps in series will be used to pump the water from the mine. The discharge from the pump station will feed a 400 mm (16 in.) diameter steel pipe, until exiting the mine, where it will change to polyethylene for the remaining length. This pump system will handle up to 120 l/s (1,900 usgpm), which will be more than adequate to dewater the mine. This rated capacity will be enough to achieve the proper draw-down to have good mining conditions. To assist in properly determining the proper draw-down rate, six, 73 mm (2.9 in.) diameter piezometer holes will be collared from the test mine lateral drift area, 30 to 137 m (100 to 450 ft.) deep to monitor static depths along the cone of depression.

Brown and Caldwell of Carson City, Nevada performed a mine dewatering study based on the conditions at the Hollister test mine. The study shows that various pumping dewatering rates for a specific number of days will result in targeted groundwater elevations needed to accommodate drying out the area as mining progresses downward. The dewatering will be 73.5 l/s (1165 usgpm), which should draw down in approximately two years.
Mine Ventilation

The primary/main ventilation system consists of two vane-axial fans operating to push used air from the mine with a range of 70.8 to 118 m³/s (150,000 to 250,000 cfm). The amount of air needed in the mine is driven primarily by the regulations required to dilute the diesel particulates generated by all of the diesel equipment which is being used within the mine at any given period.

This amount of fresh air flow will be more than adequate to dilute the diesel engine exhaust and explosive blasting fumes and to meet today's requirement of fresh air per man in the mine. However, there are new requirements for respirable diesel particulate matter (DPM) that will be promulgated in May 2008 that will require special attention. The new regulations have reduced the allowable DPMs from the original 700 micrograms/m³ to 160 micrograms/m³ by May, 2008. A combination of several methods (biodiesel fuel, exhaust filters in series with the catalytic scrubbers, low sulphur diesel fuel, and sealed enclosed cabs) and more air quantity will be needed to reduce the DPM levels to the new MSHA requirements.

The two primary ventilation fans are Spendrup 1.8 m (71 in.) 112 kW (150 hp) fans, operating at 1180 rpm with variable pitched blades. The fans are capable of being operated at much higher speeds should more air be needed in the future. This fan velocity was chosen to reduce the noise level for the fans.

One fan will be installed at the bottom of the East Alimak raise and exhaust up the raise to the atmosphere. Another fan will installed at the bottom of the West Alimak raise and exhaust up the raise to the atmosphere. The pressure gradient will draw fresh air down the Main Decline.

The East Alimak raise is 2.4 m by 2.4 m (8 ft. by 8 ft.) and will double as an escape route. The West Alimak is 3.6 m by 3.6 m (12 ft. by 12 ft.), and it will also double as an escape route. In combination, these raises are capable of exhausting up to 283 m³/s (600,000 cfm) should the need arise driven by the new MSHA requirements. Agapito Associates, Inc. have confirmed that this amount of fresh air flow will be the maximum required to dilute the diesel engine exhaust and blasting fumes to the new MSHA requirements of 160 micrograms/m³.

The auxiliary stope ventilation system consists of auxiliary fans, which pull air out of the main mine air stream and provide it to the working stope areas and development heading where most of the mine workforce and equipment are operating. The number and location of the stoping and development faces that need to be ventilated is a dynamic condition which changes often.

The auxiliary fans selected are Spendrup 1.01 m (40 in.) vane-axial fans with 44 kW (60 hp) motors that operate at 1760 rpm. Like the larger fans, they operate at a low velocity to reduce the fan noise. If more ventilation is needed, higher speed and larger horsepower
motors can be installed to operate at 3600 rpm. The fans will sit in the main east-west lateral drift and discharge into a rigid 1.37 m (54 in.) diameter duct. Where they are needed, a second fan in series will also be set, pulling from the 1.37 m diameter rigid tubing and discharging into 1.01 to 1.21 m (40 to 48 in.) diameter flexible tubing. In some cases, 3600 rpm fans will be needed for the smaller tubing.

The total air needed in each of the cross cuts that lead to the stoping areas will be dependent on the number of pieces of diesel equipment operating within that general area. At present the design quantity for each stoping area is 11.8 m³/s (25,000 cfm) and each development area is 16.52 m³/s (35,000 cfm).

Mine Compressed Air System

The mine compressed air system is located on the surface, but primarily services the underground mine. The average predicted consumption of mine compressed air to run the jackleg drills for roof bolting and the heading mud-pumps has been determined to be between 0.85 to 1.04 m³/s (1800 to 2200 cfm). The test mine facility presently has two 0.708 m³/s (1500cfm) rotary-screw compressors on site, for a total plant capacity of 1.41 m³/s (3000 cfm). These are 1983 machines which have been rebuilt and have been performing well for several years.

However during maintenance down time, a third compressor will be needed. A new or used similar compressor will need to be purchased and there is adequate space in the existing building to house the third compressor.

All compressors will feed into an existing 4.35 m³ (1150 usg) air receiver tank, which is equipped with an automatic water condensate drain valve, safety pressure pop-off valve, and pressure gauge. The delivery pipe going underground is a 150 mm (6 in.) SDR polyethylene pipe and will continue to service the mine adequately.

Mine Power

The electrical distribution for the Hollister underground mine is typical of most underground mines. Power distribution from the main transformer will be stepped down from the incoming voltage to 13,800 V. From the main transformer, power will be carried by underground cable to the portal and hence underground to the mobile load centers (MLC). Each MLC will step the power down from 13,800 V to either 4,160 V or 480 V. Most underground equipment operates on 480 V. It is estimated that the mine will have up to 8 MLC's operating when the mine attains full production.
Acknowledgements


