Ventilation Benefit Analysis for Canadian Mines
Fuelcell Loader Project

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EXECUTIVE SUMMARY

To determine the potential ventilation savings as a result of replacing diesel powered equipment with fuelcells, six Canadian underground operations ranging from a large-sized deep metal mine to small/medium-sized precious metal mines were surveyed and their current and future ventilation system modeled by means of mine ventilation programs. The potential cost benefits have been determined using ventilation simulation techniques.

These analyses show potential for 24 – 53% reductions in electrical operating cost of the primary ventilation system for associated mine intake airflow reductions of 9%, with workings located below 2400 m depth and intake airflow reductions of 24%, with workings located at relatively shallow/medium depth (between 400 – 800 m depth). Reductions in the electrical operating costs of the auxiliary ventilation system are in the order of 11 – 20%. These savings are much less than those experienced within the primary ventilation system mostly due to a limited flexibility of the auxiliary ventilation system (i.e. lack of variable speed drives and ventilation controls) to accommodate reduced airflow regimes.

Heating cost reductions as a result of replacing diesels with fuelcells were the same as the average mine intake airflow reductions, i.e. between 9 – 24%. After factoring in the relative magnitude of the primary ventilation, auxiliary ventilation and heating/cooling costs, the combined savings were between 20% under deep mining conditions and 38% with workings at shallow/medium depth.

These analyses also show reductions in greenhouse gas emissions between 27 to 38% as a result of replacing diesels with fuelcells, or between 363 tonnes of equivalent CO₂ at a small-sized precious metal mine to 8,749 tonnes of equivalent CO₂ at a deep underground mine which has to heat the intake airflow in winter and use a refrigeration system for some of the deep working areas in the summer. Even in those mines where the ventilation economic advantage is relatively small, it must be remembered that fuelcells with zero emissions will be extremely beneficial to the health and safety of the underground workforce through the elimination of diesel contaminants. Economical, workforce health and safety benefits and their potential to address increasing global environmental concerns can make the fuelcell technology an extremely attractive alternative within the mining industry.

On the other hand, heat and existing ventilation management issues are qualifiers to the potential benefits. Similarly, the need to maintain minimum airflow velocities within the working areas (and even the inactive areas) can limit environmental and cost benefits. In addition there may be other mine specific qualifiers, such as inherent dust conditions, shaft and heating system velocity limits, which are characteristic to certain underground operations.

As a result, it is due to the relevance of such qualifiers in Canada that mines have to be evaluated on an individual basis and the analyses presented in this report may not truly reflect the impact of the whole metal mining industry. For a more accurate impact analysis of fuelcell technology the majority of Canadian underground mines would need to be analyzed on an individual basis.
1. INTRODUCTION

This ventilation study comprises three parts. First, is an introduction to the potential ventilation benefits upon the introduction of fuelcell-powered vehicles to underground mining in Canada. This part also provides a background of the need for ventilation, some of the design criteria, how current air volume demands are changing and why, plus how fuelcell powered vehicles may change the design and demand. Also included in this part of the study is a classification of all Canadian underground metal mines with respect to tonnage mined, depth, mining method, equipment, haulage, ventilation infrastructure, etc., with a view to define characteristic mine types by design criteria such as heat or diesel, bulk or selective mining, size of diesel fleet and individual units. This classification will ultimately be used to help define which type and number of mines would benefit most from the introduction of fuelcells.

Second, detailed analyses of selected mines such as a large and deep base metal operation using large diesel equipment, medium sized precious or base metal mines using medium sized diesel equipment and a precious metal mine using small diesel equipment. Additional variants would be whether mobile diesel powered or tracked electric equipment were the primary haulage method and also the differing provincial diesel regulations that dictate air volumes. The analysis of these mines serves to show the varying benefit of introducing fuelcells depending on such qualifiers as heat issues, production schedule and the relative contributions of heating and primary and secondary ventilation systems in total ventilation operating costs.

Third, based upon the detailed mine analyses, and the mine classification this report will try to determine the potential benefits of using fuelcell-powered equipment in underground mines. This part will also discuss any mitigating qualifiers that could apply to individual mines, and if possible produce projections of cost, energy and environmental benefits (e.g. reduction of greenhouse gases) across Canada.

2. VENTILATION DESIGN IN UNDERGROUND MINES AND THE IMPACT OF FUELCELLS

The introduction of fuelcell-powered vehicles to the underground mining industry in Canada could be very beneficial where it replaces the current use of diesel-powered equipment. The benefits could include:

- A cleaner work environment for underground personnel to breath due to the elimination of diesel engine exhaust contaminants, some of which are suspected carcinogens. This would be beneficial to all Canadian mines but would be difficult to quantify due to the time delay before any changes in respiratory effects are observed in the workforce.
- Reduced ventilation operating costs where the volume of air sent underground was dictated by the diesel exhaust dilution requirements.
- A smaller heat load addition to the work environment because fuelcells are mechanically more efficient in their use of power. This would be beneficial where the machinery heat input is still a significant part of total heat load in mines where heat management with ventilation is the controlling design factor.
Mine’s having a smaller global environmental impact in terms of greenhouse gas emissions resulting from reduced diesel and heating fuel consumption and reduced power requirements for the ventilation systems.

These later three benefits are the focus of this study and within these benefits, the potential for cost savings in operating mine ventilation systems is very attractive to the industry because the ventilation costs can be significant and have generally been increasing. However, to assess the savings it is essential to understand the design criteria for ventilation systems, and how such systems are laid out with respect to mining methods. Consequently, prior to evaluating the potential benefits of fuelcells, a brief review of the Canadian underground mining industry, with respect to ventilation issues and the challenges of providing this essential service, is required. However to simplify this review, it can be assumed that fuelcells when introduced would be on the primary production equipment, namely load-haul-dump vehicles (LHDs or scooptrams) and trucks, and even then only on the most common vehicles. The wide scale introduction of fuelcells to the service vehicles in mines is less likely due to their diversity but could occur through adoption of such vehicles from the more commercial surface market.

The Canadian underground mining industry is heavily mechanized extensively using diesel powered or electric equipment to produce and transport ore. For the most part the Canadian industry can be subdivided into two underground mining categories “soft” rock, typified by the extraction of sedimentary deposits such as potash, salt and coal, and “hard” rock which encompasses the extraction of base and precious metal ores, the majority of such mines, but could also include uranium and diamond mines when they start to go underground.

Within the soft-rock category, the majority of the underground mines are for potash where the production and transportation is done almost exclusively with electrical equipment and diesel equipment is only used for support purposes such as personnel transport and service functions, however an exception to this is where diesel powered equipment is required to place mill tailings back within the mine. Consequently, due to the predominance of electrical equipment in the production process, the air volumes sent underground per tonne of mined material are relatively low when compared to hard rock mines. Therefore, at this time fuelcell powered vehicles would probably have limited application in potash mines due to their limited application and ventilation benefit.

In salt mines, large diesel equipment is now routinely used in the primary production and transport processes. However, the number of these mines is small, their equipment is larger than that used in most hardrock mines, and the volumes sent underground already result in low air velocity ventilation systems in their large openings. Consequently, the introduction of fuelcells to salt mines may also be limited at this time.

In Canada’s only underground coal mine, although employing some diesel equipment, ventilation rates are primarily dictated by methane dilution. Hardrock mines therefore appear to be the mines in Canada that could benefit most from the introduction of fuelcells and this is the primary focus of this evaluation.
2.1 The Ventilation of Hardrock Mines in Canada

In mines, ventilation serves three primary functions: it is needed to supply oxygen for humans, and for the combustion process within diesel engine powered vehicles; it is required to remove and/or dilute pollutants generated from the strata or the mining process; and lastly, it must supply a suitable thermal environment for workers and machinery.

Looking at the underground metal mining industry, three trends can be observed: mines are getting deeper; production is becoming increasingly mechanized and/or automated; and health standards for the underground personnel plus environmental concerns are becoming more stringent. To varying degrees these trends challenge the provision of ventilation in a mine, and to ensure the feasibility of the industry it is essential to understand why ventilation is required.

Canada has six metal mines that are planning production 3,000 m below surface. Some of these are extensions of current mining operations and others are new mines. Consequently, the ability to mine at depth is becoming an increasing concern to maintain the viability of the Canadian industry. At depth, the provision of ventilation due to its associated capital and operating costs is always a concern and one of the potential limiting factors. Hence the considerable time and effort put into the design and commissioning as evident for the documented upgrades of both Inco’s Creighton mine [1] and Falconbridge’s Kidd Creek mine [2,3].

2.2 The Effect of Depth on Ventilation

Depth can affect the economics of ventilation in three ways. Firstly, as the distance that ventilation must be supplied lengthens the associated cost in the primary ventilation system (that which delivers it underground) can increase linearly. This can be demonstrated through a basic airflow relationship:

\[
\text{Pressure Loss} = \text{Airway Resistance} \times \text{Air Quantity}^2, \text{ or simply } P = R.Q^2
\]

In this equation the pressure loss through the ventilation system is what the fans must overcome to generate the desired airflow. And generally, all other things being equal, it is the length of the system that will dictate the pressure loss because resistance is defined by:

\[
\text{Resistance} = \frac{\text{Friction Factor} (k) \times \text{Circumference} (C) \times \text{Length} (L)}{\text{Area} (A)^3}, \text{ or } R = \frac{k.C.L}{A^3}
\]

Furthermore, the power required by the fans, which is generally proportional to the operating cost, is defined as:

\[
\text{Air Power (AP)} = \text{Pressure Loss (P) \times Air Quantity (Q), or } AP = P.Q
\]

Upon substitution this becomes:

\[
\text{AP which approximates to operating cost} = R.Q^3
\]
So this equation not only shows cost is linearly proportional to resistance and hence distance, or depth for a vertical mine, but more importantly cost is a function of the cubic of the quantity.

Secondly, depth can affect the ventilation economics because it can result in increased leakage within the system. In a large ventilation system, if a mine has to supply 10% additional airflow to combat leakage as compared to a smaller mine, the resultant power/cost will be 33% greater.

Thirdly, and most importantly for deep mines, is that with depth air temperatures in a mine will increase as a result of auto-compression of the air, whereby the air becomes more dense, and due to the additional heat transfer from the strata as it gets closer to the earth’s molten core. This is unavoidable unless ameliorative measures are taken. To combat this heat gain, plus provide the same capacity for the removal of machine heat, the airflow in deep mines has to increase with depth. For example, in a Canadian mine already operating at over 2,000 m below surface an additional 300 m depth increases the air volume required by 20% and according to the cubic relationship these deeper areas would be 73% more costly to ventilate. So it can be easily seen that when heat management starts to become a concern the ventilation costs start to increase dramatically.

2.3 Heat Sources in Metal Mines

The four major heat sources in underground metal mines are:

- Mining equipment,
- Strata (geothermal gradient),
- Auto-compression: the conversion of potential energy to thermal energy as air descends through downcast shafts and vertical openings, and
- Explosives.

The Mining Equipment

Vehicles operating in access and development openings, electrical transformers and fans are all devices that convert an input power, via a useful effect, into heat. Nearly all the energy consumption of machinery underground adds heat to the mine air, since the power losses and most of the work done are converted directly or indirectly through friction into heat. In mines, greater machinery utilization and increasing power demands have resulted in underground equipment being considered to be a major source of heat. For electrical equipment, the total heat produced is equivalent to the rate at which power is supplied. In comparison, the internal combustion engines of diesel equipment have an overall efficiency of only about one-third of that achieved by electrical units. Hence diesels will produce nearly three (3) times more heat than electrical equipment for the same mechanical work output [4]. Approximately one-third of the heat generated by the diesel-powered equipment appears as heat from the radiator and machine body, one-third as heat in the exhaust gases and the remaining third as useful shaft power, which is then converted to heat by frictional processes.

A major difference between diesel and electrical equipment is that the diesel produces part of its heat output in the form of latent heat. Each litre of diesel fuel that is consumed produces approximately 1.1 litre of water (liquid equivalent) in the exhaust gases. This may be multiplied
several times over by the evaporation of water from cooling systems. In situ tests have shown that the factor can vary from 2 to 10 litres of water per litre of fuel [5].

**The Geothermal Gradient**

When air passes through an airway, its temperature usually increases. This is caused by natural geothermal heat being conducted through the rock towards the airway, then passing through the boundary layers that exist in the air close to the rock surface. This heat transmission is greater in working areas with the newly exposed strata, where the rock surfaces are often warmer than the air. However, those surfaces will cool with time until they may be only a fraction of a degree centigrade higher than the temperature of the air. This is a result of the rate of heat transmission to the air being greater than that being conducted through the rock.

The amount of heat transferred to the air from the strata is also a function of depth, as the heat of the strata normally increases with depth or increased proximity to the earth’s core. The increase of strata temperature with respect to depth is known as the “geothermal gradient”. In practical utilization, it is often inverted to give integer values and is then referred to as the “geothermal step”. As one progresses downwards through a succession of geological strata, the geothermal step will vary according to the thermal conductivity of the local material. For example, the age of the rock, its thermal properties, and its proximity to recent igneous activity can all affect the geothermal gradient. Even within a single mining district, it can vary and seldom can be considered a constant.

If the airway is wet, then the increase in dry-bulb temperature is less noticeable, the temperature may even fall as a result of the cooling effect of evaporation. In this instance, heat is still emanating from the strata but much of it is utilized in exciting the water molecules to the extent that they leave the liquid phase and form water vapour. In combination, the heat content of the air-vapour mixture has still increased through the addition of strata heat because of the internal energy of the added water vapour [6].

Usually the air temperature in downcast shafts varies in relation to the surface climate. However, it is modified by the addition of strata heat or removal of heat from the air depending on which is hotter. Within the body of the mine, the air temperatures will tend towards stable values and in the upcast shafts or raises may remain constant throughout the year. This is because the initial temperature of the intake air will determine the heat flow from the rock, and as the temperature of the air approaches the natural temperature of the rock, such heat transfer will decrease and ultimately reach a steady state related to the volume of air flowing and the thermal conductivity of the rock.

In an active area of the mine (development face, stope) due to the local addition of machine heat, the temperature of the air may be greater than the local strata temperature. In that case, heat will pass from the air to the rock and the air will cool, approaching equilibrium when its temperature equals that of the strata [6]. This heat however could be transferred back into the ventilating air upon the cessation of the mining activity.
Auto-compression

Air entering the mine through a shaft is compressed and consequently heated as it flows downward. Auto-compression occurs when potential energy is converted to thermal energy. If there is no interchange in the heat or moisture content of the air in the shaft, the compression occurs adiabatically, with the temperature raise following the adiabatic law [7]. Precise calculation of $\Delta T$ due to auto-compression is challenging, mainly because of the non-adiabatic airflow that usually prevails in mine shafts and vertical openings. Pickup of heat from the surrounding strata and moisture from the wall rock surface can both occur. Auto-compression may also be masked by the presence of other heating or cooling sources located in or near the shaft, such as air and water lines, and electrical facilities.

Explosives

Since over 75% of the energy released by the detonation is liberated in the form of heat, blasting can be a significant source of heat. The heat generated during the blasting process will dissipate as follows:

- Part of the heat will appear in the blasting fumes, which may cause a peak heat load onto the ventilated air.
- The remainder of the heat will be stored in the broken rock. The amount of heat stored in the broken rock will depend upon the mining method employed, particularly upon the quality of fragmentation. With a computer-controlled blast, as much as 40-50% of the heat produced by the explosive may be removed rapidly along with the blasting fumes.

2.4 The Impact of Mechanization

Mining like any other industry has gone from primarily hand-tool mining through the introduction of manned electrical and diesel powered equipment, namely mechanization, and is now heading towards remote controlled and semi-autonomous machines, which is automation. This change has also challenged ventilation. When diesel equipment started to be introduced into mines in the 1960s, the equipment was comparatively small with engines of less than 75 kW (<150 hp) and their number few. Since that time the number and/or total installed power of such units has dramatically increased. Initially, it was in the primary production fleet such as LHDs, drills and trucks, but a more recent trend in Canada is that for workforce mobility, the majority of underground personnel now have access to a vehicle.

Within the production fleet, the size of vehicles and their diesel engines have increased, in Canada there are now 9.2 m$^3$ bucket capacity LHDs with 260 kW (350 hp) engines being used at depth. However, these are not the largest vehicles underground, one metal mine employs at depth a 490 kW (650 hp) diesel haulage truck, and in shallow soft rock mines even larger equipment of >550 kW (735 hp) are not uncommon. However, there has been a trend in bulk mining, whereby the actual numbers of production vehicles decreases with increased capacity.

For personnel carrier vehicles, in Canada, the situation is very different. There has not only been an increase in numbers to the extent that such vehicles can now represent 50% of a mine’s vehicle population, but also the size of the vehicles’ engines have dramatically increased. Initially small engine vehicles were the norm such as 55 kW (73 hp) Toyota Landcruisers,
however mines have been replacing them with gradually larger units and today a 100 kW (134 hp) Landcruiser is more typical. Some mines have also started to use commercial highway vehicles; this has led to 160 kW (215 hp) personnel carriers and utility vehicles.

In ventilation design, the air volume supplied in mechanized mines is often based upon the need to dilute diesel exhaust contaminants and such criteria as 0.063 m³/s air/kW diesel (100 cfm/bhp) is routinely employed [8]. As with greater depths, it can be seen that increased mechanization and workforce mobility has required mines to supply more and more air underground. Again this increased air volume dramatically increases the required power and resultant cost to operate the ventilation systems.

### 2.5 Health and Environmental Considerations

Ventilation is required for the dilution and removal of contaminants, which provides a safe atmosphere for personnel to breathe. The most common contaminants are:

- strata gases, these include methane which is flammable, and radioactive radon and thoron;
- process gases, these routinely include the by-products of combustion from diesel engines and fumes from explosives;
- mineral dust, created from blasting and attrition of the rock, liberated during the blast and throughout the mineral handling chain; and
- non-mineral dusts such as soot, another by-product of combustion from diesel engines.

For the majority of Canadian hard rock mines, flammable and radioactive gases are not a significant issue and dust for the most part can be controlled at source through wetting practices or mechanical removal. The bulk of the blast fumes are generated when the mine, or area therein is not occupied, and should have been easily flushed from the workplace by the ventilation system prior to re-entry. Consequently, blast fumes are not normally a major concern. So the primary concern for Canadian metal mines is the control of by-products of diesel combustion and due to the volumes of air currently supplied to dilute these by-products for the most part all other contaminants are also controlled.

In dieselized mines there are two types of ventilation requirements. Firstly, there is a dilution requirement of the raw exhaust, and secondly, the regulations pertaining to personnel exposure stating average threshold limit values for a shift (or “TLVs”). In Canada, the dilution requirements vary widely according to the provincial jurisdiction [8]. Where specified, at the least, it could be 0.045 m³/s/kW (71 cfm/bhp) where multiple engines are used, and then they range through various values to a maximum of 0.092 m³/s/kW (145 cfm/bhp) for non-certified engines. Also, the regulations could be either a common fixed value regardless of engine type and fuel used (e.g. 0.063 m³/s/kW, 100 cfm/bhp), or a specific volume for a particular engine with a certain quality fuel (sulphur content) as a result of engine certification according to CAN/CSA [9] or MSHA [10] criteria.

Where certain Canadian provinces have adopted the certified engine approach to determine ventilation rates, this has allowed mines to take advantage of clean engine technology, such as electronically controlled engines and low sulphur fuels to alleviate the air volumes required
underground. For such engines and fuels, CAN/CSA flow requirements of 0.032 to 0.044 m³/s/kW (50-70 cfm/bhp) are not uncommon [11]. At Barrick Gold’s Bousquet II mine in the province of Quebec, this certified engine approach meant that the mine would not have to increase its ventilation capacity and its associated operating costs as it went deeper.

With respect to personnel exposure limits, some of these are met through the dilution requirements of such encompassing relationships as the EQI/AQI that is used in the CAN/CSA standard [9]. However, over the last 10-15 years, a solid fraction of the exhaust, diesel particulate matter, due to the suspected carcinogenic nature of some of its components has become a concern. The Canadian ad hoc Diesel Committee’s suggested DPM limit of 1.5 mg/m³ in 1990 that was subsequently adopted by several provincial jurisdictions was the first diesel soot regulation in N. America. However, of possibly more concern is the pending legislation in the United States that is currently under review [12]. MSHA originally proposed an interim 0.40 mg/m³ limit for total carbon (TC), which constitutes about 80% of DPM, followed by a final limit of 0.16 mg/m³ TC. Although this could mean that a much cleaner environment has to be supplied, it cannot be addressed by ventilation alone and other control measures at source or technologies such as fuel cells will be required.

The next area of concern is the global environment. Here, like all industries, mining is responsible either directly or indirectly for the production of greenhouse gases (GHGs). In mines, direct sources include diesel engines, explosives, air heating fuels and strata gas; indirect sources are through the use of electricity that has been generated from carbon-based fuels. According to “Canada’s Energy Outlook 1996-2020” the mining industry’s energy requirements have increased since 1996 and will continue to increase through to 2020 at a greater rate than the rest of the industrial sector [13]. As an increasing energy user, the mining industry will also be an increasing producer of GHGs. In 1997 the Canadian government’s position at the Third Conference of Parties on Climate Change, Kyoto Japan was to reduce GHG emission to 3% below 1990 levels by 2010 and then by a further 5% by 2015. The Canadian government’s current position on GHGs is to reduce emissions to 6% below 1990 levels by 2010, consequently energy usage and alternate energy forms will have to be addressed industry wide.

With respect to energy usage, ventilation in Canada accounts for approximately 40% of the consumption in a mine primarily through the use of electricity for fans and through fossil fuels for heating. The other main electricity user is compressed air systems that typically account for another 40%. Within the fans, auxiliary ventilation systems used to direct the air to the required workplaces, as opposed to the primary system, which delivers the air underground, can account for up to 50% of the electricity consumption. The proportion of energy used for ventilation could increase further where mines also had to refrigerate their air. Therefore if mines cannot eliminate the use of compressed air, energy reductions would have to be achieved in their ventilation systems.

So summarizing the drivers of ventilation, increased need to remove heat, larger diesel equipment and more stringent exposure regulations, historically have meant more air and increased power, cost and GHG emissions. It is because of these elements forcing up the need for ventilation that makes the potential introduction of fuel cell-powered vehicles so attractive because of their need for less air due to their zero toxic emissions.
2.6 Ventilation Design – Current Practice

In order to find potential savings in ventilation, it is necessary to understand how ventilation systems are designed and operated. Historically, mine ventilation systems were designed upon peak production demands based upon diesel or heat criteria, and are operated at this maximum level regardless of the true demand. A common scenario that still exists today is to continuously ventilate every potential working level/area with air from surface, which is then distributed through auxiliary systems. However this is changing with the gradual introduction of mine ventilation automation/optimization, and the “ventilation-on-demand” concept where air is supplied only to specific locations when necessary, and then at the appropriate volume. Through ventilation automation and the demand concept, the number of places to which air is supplied, its duration and volume can all be minimized. Consequently, the average and even total flow through the system can be reduced and significant cost and power savings achieved. Consequently, any benefit that fuelcells generate could be very different depending on whether the mine’s ventilation system is already optimized.

When using diesel exhaust dilution or heat management to determine ventilation demands, the final volume required can be dependant upon the provincial jurisdiction governing the mine and how the mine ventilation system is laid out as in the following examples.

First, for diesel exhaust control as already discussed, within provincial regulations across Canada [8], there can be as much as a 2:1 difference between the required flow for a piece of equipment at a given engine size. This can depend either on whether old potentially “dirty” or new “clean” engines are used, or on whether the CAN/CSA certification air quality approach as opposed to a fixed requirement are employed.

Second, for heat control, the number and location of fans within the system can also be significant. At the Kidd Creek mine [14], it was estimated that continuing with the mine’s existing push-pull ventilation system with fans on both the intake and exhaust side of the mine, would ultimately require 25,250 kW of fan power. This would increase the heat load within the system, raising wet-bulb temperatures by 2-3°C and dry-bulb temperatures by 8-12°C. On converting to an exhaust system, it was estimated that ultimately only 8,700 kW of fan power would be required. In addition, this lower heat load would be downwind of mining operations. Therefore, depending on the proximity of the fan heat source, less air could be required to maintain the same working environment.

So with respect to the introduction of fuelcells, the ventilation benefit could be very different depending upon the presiding diesel regulations and the overall ventilation design for heat management.

2.7 Ventilation Design – Future Practice with Fuelcell Powered Vehicles

Currently, most mine ventilation systems in Canada are designed around the total installed engine power of their diesel equipment fleet, and as a consequence of this the air supplied is typically sufficient to dilute and remove other contaminants such as mineral dust, blast fumes,
and heat issues in shallow underground mines. The only contaminants that may be inadequately covered are flammable strata gases such as methane and radioactive components as occurring in uranium mines.

In Canada’s non-coal mines, methane is known to occur, but for the most part its generation is small and localized and is an issue resolved by local auxiliary ventilation rather than the primary ventilation system. When fuelcell powered vehicles are introduced, the concern for methane would remain local to where it is produced. However, where these areas are currently ventilated for diesel equipment, reducing the level of ventilation may become an issue if air velocities at potential sources drop below 1-2 m/s permitting methane layering to occur.

For Canada’s uranium mines, the dilution and removal of radioactive components is the overriding design issue. Consequently, the introduction of fuelcells to this sector of the industry may have limited economic benefit.

As previously stated it is Canada’s hardrock mines, upon the exclusion of uranium mines, which could benefit the most from the introduction of fuelcells. How the ventilation design criteria could change within these mines, will be dependant upon the degree of conversion to fuelcell-powered vehicles. Fuelcells produce “zero-emissions” that are not harmful to humans. Consequently, they do not require any air to dilute their exhaust. This is similar to an investigation of tele-remote mining from surface with conventional diesel powered equipment, where the dilution of exhaust contaminants in the absence of humans became a minor issue. With both the fuelcell technology and the tele-remote mining some air would be required to supply oxygen. For tele-remote mining, the removal of heat from the machinery to avoid hydraulic alarms became the design criterion. For fuelcells, heat rejection may similarly be an issue but other historical concerns for the workforce, prior to the introduction of diesels can re-emerge, namely dust and blast fumes. Consequently, the design criteria may become independent of the size of the fuelcell-powered vehicle.

In deep Canadian mines, where heat is an issue, the introduction of fuelcells will not dramatically change the design criteria. What will change is the heat input from the mining machinery, which could moderate the volumes required to remove heat from all sources.

Another important aspect is the operational layout of the mine and how production areas are ventilated as this can dictate an equipment operator’s relationship to any generated contaminant [14]. For example, in some mines an LHD operator when loading may effectively always be in fresh air and any contaminants generated are rejected to a return air system; in other mines the LHD operator could be in a captive area and exposure is unavoidable. Under the “through” flow regime, reductions in ventilation with fuelcells should not change any dust exposure or fume clearance for the workforce. However, in “captive” areas where the ventilation is supplied through auxiliary ventilation systems, any reduction in ventilation with fuelcells could impact on dust concentrations and fume clearance, to the detriment of personnel exposure and productivity.
3. CLASSIFICATION OF CANADIAN HARDROCK MINES

3.1 Mining Methods

Canadian Hardrock Mines generally utilize the following mining methods:

1. Room-and-Pillar
2. Sublevel Stoping (Blasthole stoping)
3. Shrinkage Stoping
4. Vertical Retreat Stoping
5. Cut-and-Fill
6. Sublevel Caving
7. Block Caving

A description of each type of mining, including applications, development, production and comments are as follows (Excerpt from “Guide to Underground Mining Methods and Applications – Hans Harmin” [15]):

3.1.1 Room-and-Pillar Mining

With room-and-pillar mining, the orebody is excavated as completely as possible, leaving sections of ore as pillars to support the hanging wall. The dimensioning of stopes and pillars depends on the stability of the hanging wall and the ore itself, the thickness of the deposit and the rock pressure. In general, the mining aims to extract the ore as completely as possible without jeopardizing working conditions.

Pillars are normally arranged after a regular pattern. They can be circular, square or shaped elongated walls, separating the stopes. The ore remaining in the pillars can be extracted by “robbing” as a final operation in the mine, but is generally non-recoverable.

Applications:

- Orebodies with horizontal or flat dip, inclination not exceeding 30°.
- Competent rock in the hanging wall and ore.

It should be underlined that the stability of the ore and hanging wall in this respect is a flexible concept. Increasing pillar dimensions and reducing stope width will compensate for poor ground conditions. Recovery is sacrificed, however, as a great part of the orebody is left to support the back. Roof bolting is a technique which will improve the hanging wall’s stability and which is used extensively in room-and-pillar mining.

Room-and-pillar is the only feasible method of mining flat deposits of limited thickness. The method is to a great extent used for mining bedded mineralizations of sedimentary origin such as copper-mineralized shales and industrial minerals such as limestone, salt, potash and coal.
Three different systems of room-and-pillar mining are discussed below:

1. The best-known system is applicable for horizontal or nearly horizontal deposits, and can also be used in inclined deposits of greater thickness. A characteristic is that the stopes are arranged with a moderately sloping bottom, allowing the use of mobile diesel powered mining equipment.

2. The second system is applicable to inclined orebodies, with a dip in the range of 20-30°. Here the stoping proceeds upwards along the dip. The gradient of the stope bottom prevents the use of mobile equipment.

3. The third system is an adaptation to an inclined orebody of the flat stoping method described above. A special layout of stopes and sequence of ore extraction result in working areas with moderately sloping floors, again allowing mobile equipment to be used.

3.1.2 Sublevel Stoping (Blasthole Stoping)

Sublevel stoping, like room-and-pillar, is a mining method where the ore is extracted and the stope left standing empty. The stopes are often very large, with the largest dimension measured in vertical direction. The method itself is applied only to vertical or steeply dipping orebodies.

To prevent the stope walls from collapsing, larger orebodies are split up into smaller, separate stopes. Between the stopes, sections of the ore are set aside to support the hanging wall. These pillars can be designed as both vertical and horizontal separations, occasionally of substantial thickness.

Under certain conditions, the ore in the pillars can be recovered. This would normally be in the final phase of mining, when an eventual collapse of surrounding rock will not affect the regular activities.

The mining in sublevel stoping is carried out from horizontal levels and sublevels at determined vertical intervals. The sublevels are prepared within the orebody at elevations between the main levels. The ore is broken by drilling and blasting from the sublevel drifts. The blasting separates a large vertical slice of ore from the orebody, which breaks up and falls to the bottom of the stope, from where it can be recovered at the main level.

Applications:

Sublevel stoping is normally used in orebodies with the following characteristics:

- Steep dip - the inclination of the footwall must exceed the angle of repose
- Strong hanging wall and footwall
- Competent ore
- Regular ore boundaries
Development:

Comprehensive development is needed for this method, which includes:

- A haulage drift at the main level underneath the stope
- Raises for development of, and access to sublevels
- Drilling drifts inside the orebody on the sublevels
- Undercut at the stope bottom
- A loading-draw point system to allow the ore to be recovered safely
- A slot raise at the end of the stope, subsequently enlarged to a full slot, to open the stope for blasting.

Production:

Today, production drilling in sublevel stoping is done either by long-hole drilling with extension drill steel, or by the large-hole blasting technique, using DTH (down the hole) hammers for drilling.

The drilling in sublevel stoping can be done considerably in advance of the ore extraction. Large sections of ore are drilled, left in place and blasted when required. The fact that the drilling is an independent operation, with a large meterage drilled from each drift, favours the application of specialized, mechanized drill rigs such as autonomous Data Solo’s.

Ore Handling:

The lower part of the stope is designed and developed to match one of the following systems for further handling of ore:

- Loading into mine (locomotive, diesels or electric) cars through chutes. Boulders are frequent in sublevel stoping and can make this system ineffective. Blasting in chutes is complicated and lowers the output.
- Slusher scrapers to draw ore from drawpoints into mine cars.
- This method gives better access and simpler handling of oversize rock.
- Drawpoint loading with overhead loaders into mine cars.
- This system is successfully practised in some well-known large mining operations. LHD loaders are an alternative to be used in conjunction with the trackless mining layout.

Comments:

The development of extension steels, special long-hole rockdrills and more recently, the large-hole blasting technique, has made sublevel stoping a method in increased popularity. The complicated and comprehensive development may be seen as a drawback but is compensated by efficient ore production. The drilling, blasting and loading operations can be performed independently of each other, offering the potential for high utilization of equipment, and a high output with few machine units and operators. High productivity can be obtained from a concentrated area in the mine.
The sublevel stoping method requires a straightforward shaping of stopes and ore boundaries. Inside the stope, everything qualifies as ore. There are no means to recover mineralizations in the wallrock. Knowledge of the geology, the ore boundaries and a careful control of hole alignment in the long-hole pattern are key factors for the successful application.

3.1.3 Shrinkage Stoping

In shrinkage stoping the ore is excavated in horizontal slices starting from the bottom of the stope and advancing upwards. Part of the broken ore is left in the mined-out stope where it serves as a working platform for mining the ore above and to support the stope walls.

Through blasting rock increases its occupied volume by about 70%. Therefore, 40% of the blasted ore must be drawn off continuously during the mining to maintain suitable headroom between the back and the top of the blasted ore. When the stoping has advanced to the upper border of the planned stope, the stoping is discontinued and the remaining 60% of the ore can be recovered.

Smaller orebodies can be mined with a single stope, whereas large ore bodies are divided into separate stopes with intermediate pillars to stabilize the hanging wall. The pillars can generally be recovered upon completion of the regular mining.

Application:

Shrinkage stoping can be used in orebodies within:

- Steep dip; dip must exceed the angle of repose
- Competent ore
- Comparatively stable hanging wall and foot wall
- Regular ore boundaries
- Ore that is not affected by storage in the stope (certain sulphide ores tend to oxidize and decompose when exposed to the atmosphere).

Development:

The development of shrinkage stoping consists of:

- Haulage drift along the bottom of the stope
- Crosscuts into the ore underneath the stope
- Finger raises and cones from the crosscuts to the undercut
- An undercut or complete bottom slice of the stope at a level of 5-10 m above the haulage drift
- Raise from haulage level passing through the under-cut up to the main level above, to provide access and ventilation to the stope.
The development of the bottom section of the stope can be simplified in the same way as for sublevel stoping; the finger raises are deleted and the crosscuts developed for drawpoint loading with mobile equipment.

**Production:**

Drilling and blasting are carried out as overhead stoping. The rough pile of ore in the stope prevents the use of mechanized equipment. Standard practice is to use air-leg rockdrills and stoper drills.

**Ore Handling:**

The traditional ore handling system in shrinkage stoping entails direct dumping into mine cars from chutes below the finger raises. Shovel loaders are more effective in conjunction with a drawpoint loading system.

**Comments:**

Shrinkage stoping was a common and important mining method in the days when few machines were employed in underground mining. Its advantage, the ore could be dumped directly into cars through the chutes, eliminated hand loading. Today, this is of little importance and the drawbacks, labour-intensive, difficult and dangerous working conditions with limited productivity, the bulk of the ore remains stored in the stope for a long period of time, have resulted in that shrinkage stoping being replaced by other methods. Sublevel stoping, vertical retreat stoping, sublevel caving and cut-and-fill mining are methods that usually can be applied under similar conditions.

Shrinkage stoping, however, remains as one of the mining methods, which can be practised with a minimum of machinery investment, still not entirely dependant on manual capacity.

**3.1.4 Vertical Retreat Stoping**

Vertical Retreat Stoping, or Vertical Crater Retreat (VCR), is a recently designed mining method used at a few locations only. The principles behind the method utilize a unique blasting technique, the crater blasting. A patent in Canada covers the Vertical Crater Retreat mining method.

The vertical retreat stoping resembles shrinkage stoping as the ore is excavated in horizontal slices, and the stoping starts from below and advances upwards. The broken ore can remain in the stope to support the walls. The blasting technique is typical for the vertical retreat stoping, and further described in the following text.

The ore is recovered at the bottom or underneath the stope through a drawpoint system, resembling the same process used in sublevel open stoping.
Application:

Vertical retreat stoping can be used for steeply dipping orebodies under the same conditions as sublevel stoping and shrinkage stoping. As the ore can remain in the stope, it will assist in the support of the hanging wall during the stoping. The demand on a stable hanging wall is therefore less pronounced than for sublevel stoping.

Development:

The development for vertical stoping consists of:

- Haulage or loading drift along the orebody at the drawpoint level.
- Drawpoint loading arrangement and undercut of the complete stope area.
- Over cut of the full stope area at the upper level of the stope.

Production:

The ore volume contained in a stope is drilled with DTH rockdrills from the over cut, downwards, to break through into the undercut. A hole diameter of around 170 mm is used. Holes are drilled parallel to each other.

The holes are charged from the over cut with special concentrated charges, placed at a fixed distance above the lower horizontal face of the stope. The loading requires special care, as it must be done from the over cut, using strings and measuring tape to ensure that the charges are properly placed. The blasting will break the lowermost slice ore to a fragmentation that can be handled by LHD loaders.

Comments:

Vertical retreat stoping is a new mining method, and practical experiences are yet limited. Vertical retreat stoping depends more on charging and blasting techniques rather than other mining methods. It is important that this phase of the operation is developed and refined at the mine to function safely. A blast that fails to break the complete ore slice could mean that part of the ore must be sacrificed.

The experiences from vertical retreat mining until now are positive. This indicates that the method in the future will increase in popularity, replacing methods as shrinkage stoping, sublevel stoping and cut-and-fill mining.

3.1.5 Cut-and-Fill Mining Method

In cut-and-fill mining the ore is excavated in horizontal slices starting from the bottom of a stope and advancing upwards. The blasted ore is loaded and completely excavated; the corresponding volume is filled with waste material. This serves both to support the stope walls and as working platform to mine the following ore slice.
The filling material can consist of waste rock from development work in the mine, which is distributed mechanically over the stoping area. In modern cut-and-fill mines, hydraulic filling is common practice. The filling material consists of fine-grained tailings from the ore dressing plant. The tailings are mixed with water and transported into the mine and distributed through pipelines. After the water has drained off a competent fill, a smooth surface remains in the stope. The filling material can be mixed with cement to produce a harder surface.

Application:

Cut-and-fill mining can be applied in steeply dipping orebodies with reasonably competent ore. Cut-and-fill offers an advantage in terms of selectivity compared to the other methods, which are applicable to similar orebodies (sublevel stoping and shrinkage stoping). It can be adapted to orebodies with irregular and discontinuous contours to recover high-grade sections and leave lower grade untouched.

Development:

The development for mining by the cut-and-fill method consists of:

- Haulage drift along the orebody at the main level.
- Raises and manway connecting with the undercut, about 10 m above the main level; when trackless mining is applied the manway is replaced by a spiral ramp with access drifts to the undercut area.
- Undercut of the stope area and installation of a drainage system.
- Raises to connect with upper levels, for ventilation and fill transport.
- “Upper” drill holes.

Two alternative systems for drilling and blasting are used. The traditional technique is to drill holes upwards in the back and blast a slice of ore in a sort of inverted benching. The method permits drilling large roof areas without interruption and blasting of large rounds. The drilling can be done with simple wagon drills. Then the stopes are filled; however sufficient height must be left up to the back for operation of the drills.

After blasting the headroom in the stope has been increased to 7-8 m. The blasting has created a ragged back, which is difficult to control from the stope bottom. To obtain safe working conditions in the stope it is then necessary to trim the back by smooth blasting, by drilling horizontal holes with jackleg drills, the driller standing on the muck pile. This is one drawback for an otherwise efficient technique.

The drilling of “flat” horizontal holes for ore production is a common technique in mines using mechanized equipment. Here, the ore is blasted with breasting technique using flat holes. The stope can be filled almost completely, up to the back, leaving a slot above the fill high enough for the ore to expand upon blasting.
The ore volume in each blast is limited to what can be drilled from the face, thus not as large as when roof drilling is applied. The drilling is done with standard drifting jumbo drills, no special drilling equipment are required.

Frontal drilling offers several advantages. Blastholes are drilled parallel to the back so that smooth blasting is integrated in the production drilling. It also offers better selectivity than vertical drilling; low-grade areas can be left in place or blasted separately. It is also possible to make a cutout from the stope and recover mineralizations in the wallrock.

**Ore Handling:**

The smooth surface of the hydraulic fill favours the use of rubber-tired equipment. The LHD loader is therefore a common machine in the cut-and-fill mine. Transport action consists of carrying the ore to the ore-pass, normally located within the stope itself.

**Comments:**

Cut-and-fill mining has a broad range of application. This is due to its selectivity - with a higher grade run-of-mine ore than most other methods - good ore recovery and adaptability to varying rock conditions. Hydraulic filling has enhanced its benefits being a simple procedure, which to a large extent can be atomized. When filling takes place, the production of ore from the stope must be stopped, this is a characteristic for cut-and-fill mining. But such stoppages are of limited duration and there are always other producing stopes in the mine to take over the production.

Cut-and-fill mining adapts well to the trackless mining layout. Simple access to stopes is provided by the spiral ramp, permitting use of mobile, high capacity machines for drilling, charging, scaling and loading. With the jumbo drill and loader alternating from one face to the other, stope production is mainly dependant on the load-transport capacity. This method has the potential for 100% mechanization combined with good efficiency and utilization of equipment.

3.1.6 Sublevel Caving Mining Method

All caving methods function on the principle that the mineralized rock and surrounding rock fractures under more or less controlled conditions. The ore extraction creates a caving area on the surface above the orebody. A complete and continuous fracturing process is important, as unsupported underground cavities run a high risk of sudden collapse, with severe after-effects on the mining operations.

In sublevel caving the ore is divided into sublevels with comparatively close vertical spacing, normally 8-15 m. Each sublevel is developed with a regular network of drifts, covering the complete ore section. In wide orebodies the drifts are laid out as crosscuts traversing the ore from a footwall drift; in narrow deposits the drifts will run parallel to the strike.

The ore volume immediately above each sublevel drift is drilled with longholes in a fan-shaped pattern. The drilling is done as a separate operation well before the blasting. Several sublevels
can be drilled before the blasting and loading starts. The blasting on each sublevel will commence at the hanging wall or the far end of the orebody and retreat towards the footwall. Ore extraction normally retreats along an approximately straight front, which means that adjacent drifts are operated on simultaneously. Sublevels below and above follow the same production schedule.

The blasting of one fan of holes breaks up a slice of ore. The ore caves into the drift, where it is loaded and transported to ore passes. The waste rock in the hanging wall breaks up continuously and caves into the void. In the drift this can be noticed during the loading as an increasing waste dilution in the loaded material. When the dilution has reached a certain proportion the loading is discontinued and the next fan blasted. A portion of the ore will remain mixed with the waste in the caved area. The dilution of the loaded material with waste averages between 10 and 35% and the calculated ore losses range between 10 and 20%.

**Application:**

Sublevel caving is used in steeply dipping orebodies and deposits with large vertical dimensions. A minimum requirement for stability ore is that the sublevel drifts shall be self-supporting, needing only occasional reinforcement. The rock in the hanging wall must follow the ore extraction in a continuous cave and the surface be allowed to subside.

The dilution with waste material and the ore losses are factors, which influence the application of sublevel caving. Due to the waste dilution the method is preferred with ores where waste rock and the ore minerals are separated easily, for instance by a magnetic process. Another situation exists where the outline of the ore is determined by a vague cut-off-grade, and the surrounding mineralization.

**Development:**

The major part of the development consists of drifting to prepare the sublevels. This is a comprehensive undertaking and as much as 20% of the ore can be extracted already during the development phase. In addition to drifts, ore passes and ramps are needed to connect the sublevels with main levels. The share of development work in sublevel caving is substantial. However, the drifting is a repetitive, regular process, which can be organized for high efficiency with modern equipment.

**Production:**

As described earlier, the drilling is done in a fan shaped pattern from the sublevel drifts. Special types of fan-drilling rigs are used, making the long-hole drilling highly efficient.

**Ore Handling:**

The ore handling in the production area consists of loading the ore, which caves into the sublevel drift, transport to an ore pass and dumping into the same. These conditions favour the LHD
technique. Sublevels are commonly designed in the mine layout for transport lengths matched to a particular LHD loader.

As in other operations in sublevel caving, high efficiency is attained in the loading - transport phase. The loader can be kept in continuous operation, by moving from one sublevel drift to another, as soon as the ore has been extracted.

**Comments:**

Sublevel caving is schematic and regular, both in layout and working procedures. Development, production drilling and loading are carried out on separate levels, each a continuous operation independent of the other. Several work areas are always available for each machine unit. Together, these factors make sublevel caving a method where machinery equipment can be introduced throughout, maintaining an efficient mining process. The mining can be compared with an industrial process.

Waste dilution and ore losses are drawbacks, which hamper the application of sublevel caving. To minimize the adverse influence of these factors, extensive research programmes have been undertaken to determine the flow of ore within the caved area. The experience from these investigations combined with the follow-up of operational results has developed sublevel caving into a method of high technical standard.

3.1.7 Block Caving Methods

In block caving the ore fractures and breaks up by itself, from internal stresses and gravity. Only a minimum of drilling and blasting is therefore required for the ore production. The word “block” refers to the mining plan whereby an orebody is divided into large sections, blocks, with a square section, and an area of several 1000 m². Each block is completely undercut; a horizontal slot in the lower part of the block is excavated to remove the support of the rock above. Gravity forces, in the order of millions of tons, act on the rock mass. A successive fracturing takes place that affects the entire block. The rock pressure rises at the bottom of the block, crushing the material to fragments that can be handled in drawpoints or by loaders.

**Application:**

Block caving is used for large, massive orebodies with:

- Steep dip or, in a massive orebody, large vertical extension
- Rock that will cave and break into manageable fragments
- Surface that permits subsidence

These specific conditions limit the application of block caving. Applications are found within iron ore mining and the mining of low-grade disseminated ores, for instance containing copper or molybdenum.
Development:

Development for traditional block caving consists of:

- A system of loading-haulage drifts underneath each block
- Ore passes or finger raises to the grizzly level
- A grizzly level where fragmentation can be controlled and secondary blasting performed
- A second set of finger raises from the grizzly level which are widened to cones at the undercut level
- Undercut of the block

This layout refers to a traditional block caving mine plan where the ore is dumped into mine cars through chutes at the haulage level. Mines, which more recently have been developed for block caving, have adapted a trackless layout, utilizing LHD loaders to recover the ore from drawpoints. The development is simplified as the boulders are handled at the drawpoints. The grizzlies are no longer needed, and the undercut can be made directly above the loading level.

All excavations underneath the mined block are subjected to high rock pressure. The cross-section of the drifts and other excavations are therefore kept as small as possible. Heavy reinforcement with concrete is often necessary. The development stage in a block caving mine is a complicated process requiring several years to start the production.

Production:

Once the undercut is completed, the ore will start fracturing and fall down the finger raises. The caving will continue as long as the material is drawn off on the haulage level. No production drilling is required. In practice, it can be necessary to assist the caving by drilling and blasting long holes in a wide-spaced pattern. Secondary drilling to treat the over-size rock is a frequent occurrence in block caving.

Ore Handling:

The traditional block caving system with grizzly level and finger raises utilizes gravity to deliver the material into rail cars. The close control of fragmentation, which is required to load through chutes, makes the grizzly treatment a labour intensive operation, and a bottleneck in production. With a development for trackless mining and drawpoint loading, the handling of oversize material is greatly simplified, and the development work reduced.

To obtain good ore recovery from a block, it is important that the entire block sinks evenly. The volume of ore loaded at the different drawpoints or chutes within the block is therefore controlled, and follows a schedule.

Comments:

Block caving is an economical mining method under favourable conditions. The comprehensive development and the time lag until production reaches full capacity are drawbacks. There are
certain hazards as the caving and fragmentation are out of the miner’s control. Hang-ups and large boulders are difficult to tackle in a fractured environment [15].

3.1.8 Narrow Vein Mining

A more recent variation on the above mining methods has been the development of “Narrow Vein” mining methods. CANMET through its Val d’Or Laboratory has been actively involved in investigating and improving “Narrow Vein” mining methods. The following is a brief description:

There is no precise definition describing a narrow orebody. The concept will be interpreted differently by mining companies and also by countries. For example, in Sudbury mining camps, narrow veins are defined as 4.0 meters or less. In Red Lake and Abitibi mining camps, 2.0 meters or less are usually considered as narrow. Often, depending on the enterprise culture, the concept of small scale and narrow vein mining are combined.

Discussions among mining engineers in Canada and other countries proved vein-type mineralization should be differentiated from large massive orebodies. In brief, for Canada and USA, a narrow orebody is identified as a mineralization less than 4.0 meters wide. These deposits are usually mined longitudinally instead of laterally (transversal). Some deposits are mined with mechanized equipment and others use conventional air driven equipment.

Mining Methods:
- Shrinkage
- Mechanized Cut-and-Fill
- Conventional Cut-and-Fill
- Long-hole with small diameter (51 mm)
- Flat mining
- Alimak mining

3.2 Classification of Canadian Hardrock Mines

A list of Canadian Metal Mines along with pertinent information is available from the Mining Journal’s 2001 Mining Sourcebook. In order to assess the benefits of replacing diesel powered equipment with fuelcell powered equipment in underground metal mines, an initial assessment of mining methods was undertaken. Upon review, it was determined that the critical factors when assessing the mining operations was not with the specific mining method employed, however it was the size of the operation, the ventilation set-up (e.g. flow-through vs. auxiliary) and the size of the diesel equipment. Generally, replacing diesel-powered equipment operating in flow-through ventilation conditions with fuelcell powered equipment would provide the best opportunity to reduce ventilation requirements and thus reduce costs. For mines with higher production utilizing large diesel equipment, the critical criteria for maintaining current ventilation levels would be heat and dust. Diesel emissions would overall have only a minimal impact on the ventilation requirements. For mines at shallower depth with lower production
utilizing small diesel equipment, the critical criteria for establishing ventilation quantities would be diesel emissions. With the introduction of fuelcell powered equipment where the only emissions would be heat and water vapour, ventilation requirements could be reduced with potential significant cost savings.

As a result of the above rationale, Canadian metal mines were classified according to whether they were Base Metal Mines (higher production with large diesel powered equipment) or Precious Metal Mines (lower production using selective mining techniques with smaller diesel powered equipment). A third category was selected: Uranium Mines, where the ventilation requirements would be based more on the clearing of “radon daughters”. A listing of Canadian Metal Mines sorted according to whether they were base, precious or uranium mines is given in Appendix 2. The following table shows the frequency of use of the different types of mining methods used in Canadian Base Metal, Precious Metal and Uranium Mines.

<table>
<thead>
<tr>
<th>Mining Method</th>
<th>Base Metal (21)</th>
<th>Precious Metal (20)</th>
<th>Uranium (1)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Room-and-Pillar</td>
<td>3</td>
<td>1</td>
<td>0</td>
</tr>
<tr>
<td>Sublevel Stoping (Blasthole Stoping) Longhole or Uppers</td>
<td>12</td>
<td>16</td>
<td>0</td>
</tr>
<tr>
<td>Shrinkage Stoping</td>
<td>2</td>
<td>7</td>
<td>0</td>
</tr>
<tr>
<td>Vertical Retreat Stoping (VRS)</td>
<td>3</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Cut-and-Fill (Mechanized)</td>
<td>8</td>
<td>7</td>
<td>1</td>
</tr>
<tr>
<td>Sublevel Caving</td>
<td>1</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>Block Caving</td>
<td>1</td>
<td>0</td>
<td>0</td>
</tr>
</tbody>
</table>

Notes:

1. Production for Base Metal Mines ranged from 1,500 - 11,000 tpd
2. Production for Precious Metal Mines ranged from 300 - 3,300 tpd
3. Production for the Uranium Mine was 450 tpd
4. The numbers in brackets refer to the number of mines.
5. The numbers in the table well exceed the total number of mines as many of the mines utilized up to 3 different mining methods in their operation (see Appendix 2).

A detailed assessment of the impact of replacing diesel equipment with fuelcell powered equipment on ventilation requirements and subsequent cost savings was undertaken for the following mines:

**Base Metal:**
- INCO Coleman/McCreedy East (Mechanized Cut-and-Fill)
- INCO Creighton Mine (Vertical Retreat)
- Aur Resources, Inc., Louvicourt Mine (Sublevel Stoping - Longhole Open)

**Precious Metal:**
- Battle Mountain Canada Ltd., Holloway Mine (Sublevel Stoping - Longhole)
• Cambior Inc., Doyon Mine (Sublevel Stoping - Longitudinal Large Diameter)
• Agnico-Eagle, Laronde Mine (Shrinkage Stoping - Open Traverse)

From the above table it can be seen that detailed assessments were conducted for the most commonly used mining methods for both Base and Precious Metal Canadian mines. The one exception would be for Mechanized Cut-and-Fill for a Precious Metal mine. This could be addressed in a future study.

4. MINE VENTILATION COST BENEFIT ANALYSIS

As previously mentioned, most mine ventilation systems in Canada are designed based upon the diesel exhaust criteria. In other words, they are designed according to the total installed engine power of their diesel equipment fleet. The air volume supplied to dilute diesel exhaust contaminants below their threshold limit values is typically sufficient to dilute and remove other contaminants generated during the mining cycle such as mineral dust, blast fumes and overcome the heat transferred from strata and mining machinery.

In order to determine potential ventilation savings as a result of replacing diesel-powered equipment with fuelcells, the following underground operations (ranging from large-sized and deep base metal mines to medium/small sized precious metal mines) were surveyed and their current and future ventilation system modeled in case that available mine ventilation simulators could be used (i.e. 3DCanvent\textsuperscript{TM}, VnetPC\textsuperscript{TM}):

1. Holloway Mine – A heavily dieselised small sized underground precious metal operation, with workings at relatively shallow depth (between 300 m – 800 m), located in northern Ontario, Canada, owned and operated by Battle Mountain of Canada.

2. Creighton Mine – A large sized and deep underground base metal mine with production extracted from the upper orebody (1,000 m - 2,100 m depth) and the deep orebody (2,100 m - 2,500 m depth), located in the Sudbury Basin, owned and operated by INCO LIMITED, Ontario Division.

3. McCreedy East Mine – A medium sized and medium depth underground operation, with working located between 3,300 m and 4,800 m depth, producing not only nickel but also copper and precious metals, located in the Sudbury Basin, owned and operated by INCO LIMITED, Ontario Division.

4. Laronde Mine – A relatively large sized and deep underground gold mine, with a new orebody located between 754 m - 2,250 m below the surface, accessed by two of the deepest single shafts of all underground operations between Rouyn-Noranda and Val d’Or, owned and operated by Agnico Eagle Ltd.

5. Louvicourt Mine – A medium sized underground copper and zinc mine, with workings located between 415 m and 910 m depth, accessed by two circular concrete lined shafts, both 910 m deep, owned and operated by Aur Louvicourt Inc.
6. Doyon Mine – producing 216,000 ounces of gold, situated on the Cadillac gold fault in the Abitibi region of the Province of Quebec, located between Rouyn-Noranda and Val d’Or, owned and operated by Cambior Inc.

Holloway, Creighton and McCreedy East mines were modeled using 3DCanvent™ and VnetPCTM mine ventilation simulators, respectively, where the potential benefits as a result of replacing diesels with fuelcells have been determined through ventilation simulation techniques. The cost benefit analysis for Laronde, Louvicourt and Doyon mines was based upon an underground survey and detailed information provided by the mines regarding the mining process and their support infrastructures.

For each of the above-mentioned underground operations, a detailed description including the mine layout, the mining process and its support infrastructures, as well as an in-depth analysis of the potential cost and environmental benefits as a result of replacing diesels with fuelcells is presented in Appendix 1.

The following sections will present a summary of the Canadian case studies including the potential cost and environmental benefits such as the mine intake airflow, operating cost and GHG emissions reductions within the primary and auxiliary ventilation system as a result of replacing diesels with fuelcells. This summary will also present logistical and other environmental considerations such as a summary of the potential mitigating qualifiers to the introduction of fuelcells into Canadian underground operations.

4.1 Total Mine Intake Airflow Reductions – Diesels vs. Fuelcells

Usually fresh air enters the ventilation system through one or more “downcast” shafts or other vertical and near-vertical airways connected to the surface. The air then flows along “intake” airways to the working areas (e.g. production stopes, development faces), where the majority of pollutants are added to the airflow. These include diesel pollutants, dust and a combination of other potential hazards including toxic and flammable gases, heat, humidity and radiation. The contaminated air passes back through the system along return airways and directed to the surface via one or more “upcast” shafts and vertical airways.

As air flows through a mine opening, it must be supplied with power to overcome the pressure losses that occur in the airflow. In section 2.2 the air power was shown to be proportional to both the pressure loss (P) and the air quantity (Q). However, the pressure loss is proportional to Q², resulting in the power being proportional to the cube of the air quantity (P \( \propto Q^3 \)). From this relationship, it is apparent that the quantity is extremely important in the determination of the air power, therefore, any attempt should be made to reduce the mine total intake airflow to its lowest possible value. This means that no excess air should be provided and leakage should be controlled in order to minimize the mine’s total intake airflow. On the other hand, sufficient air volumes are needed in the active areas of the mine in order to dilute and remove dust, blast fumes, diesel exhaust components and other pollutants below their threshold limit values.
In Canada, provinces and territories are responsible for occupational health and safety in mines with the exceptions of Crown Corporations and uranium mines that fall under the federal jurisdiction. The preparation of Canadian standards for application to diesel equipment in mines began in the late 1970’s and was an initiative of the provincial inspectorates in forming a Canadian Safety Association (CSA) Standard Committee with representatives from all parts of the industry. Two standards resulted from this work, published by the CSA, one for non-gassy underground mines (CSA 1990), and the other for gassy underground coal mines (CSA 1988). Both standards describe the technical requirements and procedures necessary for design, performance and testing of new or unused diesel-powered machines in underground mines. The main feature of the CSA Standard is the application of a quality criterion by which to assess the comprehensive toxicity of diesel emissions in order to effectively prescribe ventilation requirements for a certified machine. The use of a comprehensive criterion is important because of the substantial changes in the individual concentrations of toxic constituents produced by different engines. Table 1 summarizes engine certification and ventilation requirements for areas where diesel engines are used in underground metal mines [8].

Table 1: Ventilation Requirements in Canada Based Upon Diesel Engine Certification

<table>
<thead>
<tr>
<th>Province/Territory</th>
<th>Certification Standards</th>
<th>NOTES</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>CSA</td>
<td>MSHA</td>
</tr>
<tr>
<td>British Columbia</td>
<td>Yes</td>
<td>-</td>
</tr>
<tr>
<td>Alberta</td>
<td>Yes</td>
<td>-</td>
</tr>
<tr>
<td>Saskatchewan</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Manitoba</td>
<td>Yes</td>
<td>Yes</td>
</tr>
<tr>
<td>Manitoba</td>
<td>Yes</td>
<td>Yes</td>
</tr>
<tr>
<td>Nova Scotia</td>
<td>Yes</td>
<td>Yes</td>
</tr>
<tr>
<td>Northwest Territories and Nunavut</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Yukon Territories</td>
<td>-</td>
<td>-</td>
</tr>
</tbody>
</table>

The current diesel exhaust design criterion in Ontario is 0.063 m³/s/kW (100 cfm/bhp) regardless of the engine type. In other jurisdictions such as in the Province of Quebec, the Canadian Safety Association (CSA) Standards that are specific to an engine type and take account of “clean-engine” technology have been adopted. As a result, for a similar operation the total mine intake...
airflow based upon the clean-engine technology can be as much as 10-15% less than the mine total intake airflow based upon the fixed diesel exhaust criterion. This means that the mine intake airflow reductions as a result of replacing diesels with fuelcells could be less significant in the Province of Quebec when compared to the Province of Ontario.

From all surveyed Canadian operations, Creighton Mine’s ventilation system was the only one whose design was based upon two design criteria, diesel exhaust within the upper orebody and heat management within the deep orebody. Within the upper orebody the mine currently uses 0.079 m$^3$/s per rated kW of diesel power in its overall primary design, which is notably higher than the Provincial requirement of 0.063 m$^3$/s per rated kW of diesel power to take account of increased leakage and heat rejection issues. For the deep orebody the mine has determined the airflow requirements through climatic modelling for each production level. Figure 1, shows the increasing intake airflow temperature and minimum airflow requirements for each particular level of the deep orebody as function of the mine’s production rate, upon taking into account the heat generated from strata and the mining equipment (diesel) during the mining cycle.

![Figure 1 - Airflow Requirements and Airflow Temperatures as a Function Of The Mining Depth](image)

Determining the appropriate ventilation requirements in a mine employing fuelcells is rather difficult because it depends upon the degree of conversion of the underground diesel fleet and how the remaining light-duty diesel vehicles and other secondary requirements are accounted
for. For shallow and medium depth underground operations where the non-equipment heat generation is not an issue, the intake airflow requirements for fuelcells were determined based upon the mine’s production rate such as 0.042 m$^3$/s (90 cfm) per tons per day (as used in the MSM analysis for blast fume clearance and dust control in an underground gold mine with no other reported design criteria) and 25% additional allowance for miscellaneous needs [14]. Further to this, the mine total intake airflow was then revised according to the airflow distribution in the ventilation system, minimum airflow velocities along the main haulage routes and inactive areas of the mine and other secondary considerations such as ventilation of crushers, mechanical shops, refuelling stations, etc.

For the deep and hot underground operation with increasing heat generation from the non-equipment heat sources (i.e. strata, auto-compression), the fuelcell-based demands have been determined by means of climatic modelling. This mine-supplied data showed that increasing volumes are required at depth to combat the increasing component of heat from the strata, although the input from machinery would tend to stay the same. To obtain the needs of fuelcells, the relative machinery and non-machinery heat components at varying depths were determined from the mine’s production rate flow requirements and knowledge of the thermal parameters within the mine. Once the relative machinery and non-machinery components were determined, a simplified analysis on replacing diesel machinery with fuelcell units was performed as follows:

An underground climatic model has been developed using Climsim™ simulator for a 206 kW diesel LHD operating in a typical dead-end development (4.9 m wide by 4.5 m high) at 2200 m depth. Climatic simulations show that of the total heat load of 392 kW added to the air, 206 kW (52%) is generated by the diesel LHD and 186 kW (48%) is generated by non-equipment heat sources. Upon replacing the diesel powered LHD by a 206 kW fuelcell powered unit, the total heat load decreases to 275 kW, of which 89 kW (33%) is generated by the fuelcell LHD and 186 kW (77%) is still generated by non-equipment heat sources. For both diesels and fuelcells the intake airflow requirements were determined such that ambient wet bulb temperature were maintained below 28ºC. Then, by varying the VRT and the barometric pressure the required intake airflows during the mining cycle have been determined for both diesel and fuelcell powered LHDs as a function of the mining depth. Figure 2 shows the relative reduction that might be possible from employing fuelcells at increasing depth as derived from the respective diesel/fuelcell airflow requirements.
Although this analysis may not be precise, it indicates that with increasing depth the heat from non-machinery sources increases. A point is reached, (at approximately 2,700 m depth) where for heat removal purposes whether a mine is using a fuelcell or diesel based equipment, makes little difference to the airflow requirements.

To deliver the required air volumes to the production areas and maintain the recommended minimum airflow velocities within the inactive areas of the mine, the mine total intake airflow and the potential intake airflow reductions as a result of replacing diesels with fuelcells have been determined by means of modelling and ventilation simulation techniques. For each particular underground operation the applied design criterion as well as any specific requirements such as minimum airflow velocities and ventilation requirements for refuelling stations and mechanical shops are described in detail in Appendix 1 – Mine Ventilation Cost Benefit Analysis.

For all surveyed mines, the mine total intake airflow and its potential reductions as a result of replacing diesels with fuelcells is presented in Table 2 as well as in a graphical format in Figure 3.
Table 2 - Mine Total Intake Airflow and Potential Reductions for Fuelcells vs. Diesels

<table>
<thead>
<tr>
<th>U/G OPERATION</th>
<th>Production Rate (tones/day)</th>
<th>Mine Total Intake Airflow (m³/s)</th>
<th>Potential Intake Airflow Reductions Fuelcells vs. Diesels (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total Intake Airflow</td>
<td></td>
<td>Diesel Powered Equipment</td>
<td>Fuelcell Powered Equipment</td>
</tr>
<tr>
<td>Holloway Mine – Battle Mountain</td>
<td>2,300</td>
<td>143</td>
<td>116</td>
</tr>
<tr>
<td>Creighton Mine – Upper Orebody, Inco</td>
<td>4,000</td>
<td>608</td>
<td>459</td>
</tr>
<tr>
<td>Creighton Mine – Lower Orebody, Inco</td>
<td>4,000</td>
<td>652</td>
<td>592</td>
</tr>
<tr>
<td>McCreedy East Mine – Inco Limited</td>
<td>4,000</td>
<td>538</td>
<td>432</td>
</tr>
<tr>
<td>Laronde Mine – Agnico Eagle</td>
<td>7,000</td>
<td>703</td>
<td>605</td>
</tr>
<tr>
<td>Louvicourt Mine – Aur Resources Ltd.</td>
<td>4,300</td>
<td>387</td>
<td>318</td>
</tr>
<tr>
<td>Doyon Mine – Cambior Limited</td>
<td>3,700</td>
<td>354</td>
<td>276</td>
</tr>
</tbody>
</table>

Figure 3 - Potential Reduction in Mine Total Intake Airflow Fuelcells vs. Diesels

Table 2 and Figure 3 show potential for 9 - 24% reductions in the mines total intake airflow as a result of replacing diesels with fuelcells. For example, at Creighton mine within the future
scenario (year 2018) with ore reserves extracted only from the deep orebody (2400-2500 m depth), due to increasing heat generation from the non-equipment heat sources, simulation exercises show potential for only 9% reduction in intake airflow compared to 24% reduction within the current situation with the majority of production coming from the upper orebody. Further to this, as shown in Figure 2, at approximately 2,700 m depth a point will be reached where due to the non-equipment heat removal purposes the replacement of diesels with fuelcells makes little difference to the airflow requirements.

With the exception of Doyon Mine, the potential intake airflow reductions at the other two operations from the Province of Quebec (Laroned - 14% reduction, Louvicourt - 7% reduction) are somehow less significant than at Creighton-Upper Orebody and McCreedy East mines, located in the province of Ontario. This is partially due to the fact that in the province of Quebec the amount of intake air volumes needed to dilute diesel pollutants have been determined based upon the “clean” diesel engine requirements, which is 10-15% lower than the airflow requirements in the Province of Ontario.

4.2 Ventilation System Operating Cost – Diesels versus Fuelcells

When considering the development of a new airway, the two important economic variables that must be considered are the cost of developing and the cost of operating the airway. These costs in general can be divided into fixed costs and variable costs. The primary fixed cost is the capital cost associated with the construction or development of the airway, which would include the cost of materials, labour, equipment and support services necessary to develop the airway. The fixed cost would also include ground support maintenance costs, which would occur whether the airway is being used or not. The variable or operating cost normally consists simply of the fan power cost necessary to move a required amount of air volume through the airway. The operating cost for this situation can be determined by calculating the power required to move the required air volume and multiplying by the unit cost of power (i.e. $/kWh) [7].

For example, a surface/underground fan unit, comprising an electric motor, transmission and impeller, converts electrical energy into air power. The air power is reflected as kinetic energy of the air and a rise in total pressure across the fan. The air power delivered by the fan can be quantified as [16]:

$$\text{Power (air)} = P_{ft} \times Q \ (W), \text{ where:}$$

$$P_{ft} = \text{rise in total pressure across the fan (Pa)}$$

$$Q = \text{airflow} \ (\text{m}^3/\text{s})$$

However, the electrical power taken by the fan motor will be greater than this, as power losses occur in the fan motor, transmission and impeller. If the overall fractional efficiency of the unit is $\eta$, then the electrical input power to the motor will be [16]:

$$\text{Power (electrical)} = \frac{P_{ft} \times Q}{\eta} \ (W)$$
Electrical power charges (E) are normally quoted on cost per kilowatt hour (kWh). Hence, the cost of operating a fan for 24 hours per day over 365 days in a year can be expressed as:

\[
\text{Operating Cost} = \frac{P_n \times Q \times E \times 24 \times 365}{1000 \eta} \quad \text{($/year)}
\]

As mines are deepening and at the same time expanding laterally, to maintain adequate ambient temperatures for underground workers can become a great challenge. Cooling of some type is necessary when ventilation alone is insufficient to maintain adequate thermal environmental conditions in deeper hotter working areas. However, refrigeration systems can represent an extremely large portion of the combined ventilation system operating cost and in some cases the cost to operate surface and underground refrigeration plants can represent as much as 50 - 75% of the combined ventilation operating cost. Therefore, every attempt should be considered in order to minimize the use of refrigeration systems.

From the surveyed Canadian mines, two operations were found to be employing refrigeration at depth. These two methods of refrigeration are very different, as follows:

1. Laronde Mine – Agnico Eagle, Quebec. At this operation a 2-unit, 1.65 MW/unit cooling system has been installed on level 146. Here, mine water is used in the condensers of the central refrigeration units. The chillers of the units are used to cool water in a closed circuit; the water is then circulated to heat exchangers located in the working areas. This operation is also planning to install another 1.65 MW cooling unit on the same level in July-August, 2003, followed by the installation of a 13 MW surface cooling plant.

2. Creighton Mine, Sudbury, Ontario – Inco Limited; Here the extreme temperature variation between winter and summer is used advantageously in providing natural air conditioning within the upper and deep orebodies. Air entering the mine in winter passes through broken rock of an abandoned open pit and two large stopes located near the surface; water is sprayed into the air and freezes, in turn warming and humidifying the intake airflow. The heat flow is reversed in summer, as the ice stope that has formed in winter melts and cools the intake airflow. With 608 m³/s intake airflow, the equivalent refrigeration is approximately 8-9 MW.

Fuelcells have zero toxic emissions, so diesel exhaust dilution and more limiting exposure regulations pertaining to specific exhaust constituents become less of an issue in ventilation design. Furthermore, fuelcells are mechanically more efficient in the regard to the amount of heat they transfer into the airflow, so during the mining cycle, their intake air volume can be reduced, if compared to same-size diesel units. Reduced airflow requirements result in decreased electrical power consumption, heating fuel usage and hence capital and operating costs.

The combined ventilation operating cost includes the primary and the auxiliary ventilation systems operating costs as well as any associated intake airflow heating and cooling costs. Within the primary ventilation system air usually moves in a through-flow manner from downcast shafts and ramp systems, across the levels, sublevels and stopes towards return air raises or upcast shafts. The airflow across each of the levels is usually controlled by doors,
regulators and booster fans. The auxiliary ventilation system consists of auxiliary fans and a ducting system used to supply air to the working faces and dead-end development headings. Ideally the auxiliary ventilation system should have no impact on the distribution of airflow around the main ventilation infrastructure.

As previously shown, the ventilation system’s operating cost is linearly proportional to the power consumption and electrical power charges, which are normally quoted on cost per kilowatt-hour. The operating cost within the primary ventilation system for fuelcells versus diesels at Holloway, Creighton-Upper Orebody, Creighton-Deep Orebody and McCreedy East mines has been determined through ventilation simulation techniques, while the operating cost of the primary ventilation system at Laronde, Louvicourt and Doyon mines was based upon detailed information provided by the operations. For all surveyed mines a detailed description about the combined ventilation operating cost and its associated costs are presented in Appendix 1 – Mine Ventilation Cost Benefit Analysis.

<table>
<thead>
<tr>
<th>U/G Operation</th>
<th>Primary Ventilation Costs 1000$Can/year</th>
<th>Auxiliary Ventilation Costs 1000$Can/year</th>
<th>Heating Costs 1000$Can/year</th>
<th>Cooling Costs 1000$Can/year</th>
<th>Combined Operating Costs 1000$Can/year</th>
</tr>
</thead>
<tbody>
<tr>
<td>Holloway Mine - Diesel Fuel-cells ($0.067/kWh)</td>
<td>207</td>
<td>116</td>
<td>188</td>
<td>166</td>
<td>576</td>
</tr>
<tr>
<td>Creighton Mine - Upper Orebody Diesel Fuel-cells ($0.055/kWh)</td>
<td>3,400</td>
<td>1,770</td>
<td>1,500</td>
<td>1,270</td>
<td>N/A</td>
</tr>
<tr>
<td>Creighton Mine - Deep Orebody Diesel Fuel-cells ($0.055/kWh)</td>
<td>4,200</td>
<td>3,200</td>
<td>1,800</td>
<td>1,600</td>
<td>N/A</td>
</tr>
<tr>
<td>McCreedy East Mine Diesel Fuel-cells ($0.065/kWh)</td>
<td>2,750</td>
<td>1,384</td>
<td>2,200</td>
<td>1,769</td>
<td>350</td>
</tr>
<tr>
<td>Laronde Mine Diesel Fuel-cells ($0.0242/kWh)</td>
<td>2,400</td>
<td>1,544</td>
<td>934</td>
<td>781</td>
<td>710</td>
</tr>
<tr>
<td>Louvicourt Mine Diesel Fuel-cells ($0.0387/kWh)</td>
<td>420</td>
<td>233</td>
<td>479</td>
<td>383</td>
<td>800</td>
</tr>
<tr>
<td>Doyon Mine Diesel Fuel-cells ($0.042/kWh)</td>
<td>550</td>
<td>260</td>
<td>1,635</td>
<td>1,330</td>
<td>320</td>
</tr>
</tbody>
</table>

Table 3 shows a summary of the primary and auxiliary ventilation system’s operating cost, intake airflow heating/cooling costs as well as the combined ventilation system operating cost for fuelcells versus diesels. Table 4 shows the potential operating cost reductions (as %) within the
primary ventilation system, the auxiliary ventilation system as well as the combined operating cost reduction as result of replacing diesels with fuelcells.

From Table 3, we can notice that with the exception of Doyon mine (excluding any heating/cooling costs) the auxiliary ventilation operating cost can represent between 30 to 65% of the combined ventilation operating cost. At the Doyon mine, the auxiliary ventilation cost represents approximately 75% of the combined ventilation costs. This is because the mine employs “series” auxiliary ventilation practice. Furthermore, at this operation most of the ore reserves are extracted using selective mining methods, with stopes usually following the high-grade narrow veins, which are relatively far from the vertical intake fresh air raises. To overcome the pressure losses along the auxiliary ducting system, the mine uses approximately 226 secondary fans with electrical motor powers ranging from 15 to 120 kW. However, these auxiliary fans are only operational when needed, therefore only 40% of them are operating in order to deliver the required air volumes to the active areas of the mine.

Table 3 also shows that presently, the operating cost of the 2-unit 1.65 MW/unit refrigeration system at Laronde mine is $340,000/year. However, the refrigeration system’s operating cost is expected to increase to $540,000/year by 2004 and to approximately $1,700,000/year with the installation of a new 13MW surface cooling plant. As a result, the combined heating and refrigeration costs at Laronde mine can then represent as much as 53% of the combined ventilation operating cost. It can be seen that with increasing depth, mine ventilation system’s operating costs can significantly increase. On the other hand, with Creighton mine employing a natural air conditioning system, the intake airflow heating/cooling costs are negligible compared to the operating cost of the primary and auxiliary ventilation system.

Figures 4, 5 and 6 show a summary of the primary, auxiliary and combined ventilation system operating costs and their potential reductions as a result of replacing diesels with fuelcells in a graphical format.

Table 4, and Figure 4 show potential for a 24% (Creighton Mine – Deep Orebody) to 53% (Doyon Mine) reduction in the electrical operating costs of the primary ventilation system. Reductions in the mine’s total intake airflow (as a result of replacing diesels with fuelcells) would mirror the heating cost reductions between 9% (Creighton Mine – Deep Orebody) to 24%.
### Table 4 - Combine Ventilation Operating Costs and Potential Reductions for Fuelcells vs. Diesels

<table>
<thead>
<tr>
<th>U/G Operation Operating Cost</th>
<th>Combined Ventilation Operating Cost ($Can/year)</th>
<th>Potential Operating Cost Reductions Fuelcells vs. Diesels (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Diesels</td>
<td>Fuelcells</td>
</tr>
<tr>
<td>Holloway Mine</td>
<td>971,000</td>
<td>749,000</td>
</tr>
<tr>
<td>Creighton Mine - Upper Orebody</td>
<td>4,900,000</td>
<td>3,040,000</td>
</tr>
<tr>
<td>Creighton Mine - Deep Orebody</td>
<td>6,000,000</td>
<td>4,800,000</td>
</tr>
<tr>
<td>McCreedy East Mine</td>
<td>5,300,000</td>
<td>3,433,000</td>
</tr>
<tr>
<td>Laronde Mine</td>
<td>4,384,500</td>
<td>3,226,000</td>
</tr>
<tr>
<td>Louvicourt Mine</td>
<td>1,699,000</td>
<td>1,271,000</td>
</tr>
<tr>
<td>Doyon Mine</td>
<td>2,505,000</td>
<td>1,840,000</td>
</tr>
</tbody>
</table>

#### Figure 4 - Primary Ventilation Cost Reductions – Fuelcells vs. Diesels

![Figure 4 - Primary Ventilation Cost Reductions – Fuelcells vs. Diesels](image-url)
Figure 5 - Secondary Ventilation Cost Reductions – Fuelcells vs. Diesels

Figure 6 - Combined Ventilation Cost reductions – Fuelcells vs. Diesels
Reductions in the electrical operating costs of the production auxiliary fans are between 11% at Creighton Mine – Deep Orebody scenario to 20% at the Louvicourt Mine (see Figure 5). These savings are much less than those experienced within the primary ventilation system mostly because the electrical motors of the auxiliary fans do not have variable speed drives (such as most of the surface fan motors) and other ventilation controls in order to accommodate variable airflow regimes, therefore, the auxiliary airflows may not be trimmed at the same level as the airflow within the primary ventilation system. Unless the mine replaces all auxiliary fans to accommodate new airflow regimes, reduced air volumes could only be obtained by throttling the current auxiliary systems at their discharge. This is the common practice at many mines, which however does not change the operating cost of the auxiliary system.

If the mines would choose to downsize their current secondary fans to the next lowest size available in their inventory (i.e. from 100 kW to 75 kW, from 75 kW to 60 kW and from 60 kW to 37 kW), their total fan power could be reduced by 12% to 20% and similar reductions in operating costs would follow.

Another important aspect in reducing the power consumption within the auxiliary ventilation system is the operational layout of the mine and how production stope s are ventilated as this can allow an equipment operator to be exposed to any generated diesel pollutants. In operations employing a “through” flow ventilation practice within the production stopes (as shown in Figure 7), an LHD operator when mucking, may effectively always be in fresh air and any contaminants generated during the mucking operation are rejected into a return air system. Further to this, under the “through” flow ventilation practice, reductions in ventilation as result of replacing diesels with fuelcells should not affect any dust exposures of blast fume clearance for the workforce.

However, in captive areas, with the mine employing “series” auxiliary ventilation system (as shown in Figure 8), any reductions in ventilation as a result of replacing diesels with fuelcells could impact on blast fume clearance and dust concentrations generated during the mining cycle to the detriment of workers health and productivity.
For example, at Holloway Mine where the ventilation design criteria is diesel exhaust and the primary consideration is blast fume clearance, with the current level of ventilation the blast fumes are cleared within 30 minutes. However, long-hole blast fumes can take in excess of 2 hours to clear, therefore, they are normally scheduled at the end of the production week for clearance across the non-active weekend. As this operation has a 3-shift per day production schedule, any increase in blast fume clearance as a result of reduced ventilation may not be acceptable. As there will be instances when long-hole blasts occur during the week, in order to clear the blast fumes in a reasonable time period the mine would need to retain the flexibility of a high volume flush. As a result the air volume within the auxiliary ventilation system cannot be trimmed at the same level as the air volumes within the primary ventilation system. As a further consequence, power and operating cost reductions within the auxiliary systems are much less than those experienced within the primary ventilation system (See Figures 4,5).

4.3 The Benefits of Future Technology

Looking back at the Canadian underground metal mining industry and its trends, increasing needs to remove heat, larger diesel-powered equipment and more stringent exposure regulations would indicate that more air volumes need to be supplied underground, which consequently will increase power consumption and greenhouse gas emissions. In opposition to this, remaining cost effective and reducing GHG emissions, through less power usage would tend to indicate that less air volumes should be supplied. The only way both these goals can be achieved is through innovation and better management and utilization of the air volumes sent underground.

To avoid prohibitive ventilation costs, the industry must try and minimize the air volumes being sent underground to what is really needed for their current mining operations and explore the potential for further reductions through the application of new technology. The preceding sections have discussed how the air volumes that are supplied underground can be minimized in order to reduce the ventilation subsystem’s operating cost by replacing diesel-powered equipment with fuelcells. In addition to the fuelcell technology, one area that has not been looked at yet, is the physical elements of the distribution network and for ventilation systems to achieve their most cost effectiveness, we must again return to current ventilation design practice to see how we can come up with a better ventilation design criterion.

4.3.1 Ventilation System Based Upon The “Life-Cycle” Airflow Demand Schedule

A typical design practice is to define what volume of air is required at some point in the future (say 10 years from now), and then size the infrastructure accordingly. For example, primary airways are often designed upon their final requirements, which could be oversized for a significant portion of the mine’s life. These airways can also be designed in isolation without considering their interrelationship with the rest of the ventilation system. This could lead to potentially oversized airways, developed at unnecessary extra expense, or proposed costs that could negate the viability of the mine. So again, better economic based models are required.

Today, mining process simulators that can display the development of a mine from start to finish are being used to test varying mining scenarios and the economics thereof. For example, such
Simulators were used by INCO to show how Telemining™ would improve the economics of a low-grade deposit on considering the accelerated extraction and increased equipment utilization that the automated process could provide.

We are currently exploring how such process simulators, which show the whole mine life could be adapted to forecast the need for materials or a specific resource, and one of these would be the ventilation. The results from these would then be used in standard ventilation simulators to determine the ventilation costs. Using such simulators, it should be possible to [17]:

- Compare the long-term life cycle requirements for the ventilation system under various mining methods, with different mobile equipment, and at various levels of automation.
- Evaluate the short-term life cycle requirements as would be supplied in a “Ventilation-on-demand” system.
- Obtain the minimum through to maximum ventilation requirements, the duration of each, and what processes are contributing to the peak demand.
- Explore options to reduce peak ventilation demand similar to electrical power management practice.
- Integrate in parallel the development of the ventilation system with mining, so heading towards “develop-as-you-go” as opposed to developing for final requirements with high upfront costs.
- Size the infrastructure according to life cycle demands as opposed to final or peak demands. For example, it may be more cost effective to have a slightly smaller fan or raise and pay the penalty for the duration the higher demand is required.

4.3.2 Ventilation-On-Demand Based Systems

In the last 5-10 years, the mining industry has recognized the importance of reducing the power consumption underground, and mines have made various efforts to control their primary and auxiliary ventilation. However, the full benefit has never been achieved and the primary cause of this has been a lack of necessary control and information gathering infrastructure. For optimal efficiency a mine requires a “ventilation-on-demand” based system (17,18,19).

CANMET-MMSL has been a long-standing proponent of “ventilation-on-demand” based systems and consequently considerable resources have been put towards developing the supporting technologies needed for its implementation. Such systems are comprised of four main building blocks (see Figure 9):

1. Vehicle Tracking and Identification System – in order to be able to direct the air accordingly it is essential to know where the primary demand is at any time. In highly mechanized mines, this means knowing where vehicles are and their identity. This requires a reliable and cost effective vehicle tracking and identification system, and it is the lack of such a technology that has been one of the hurdles to the implementation of the demand concept. Depending upon the degree of control to be achieved i.e. mine-wide as opposed to local, the actions prompted by such a system may also have to be evaluated through ventilation simulators to ensure their validity and safety.
2. Ventilation Controls – in a ventilation system there are two types of control, active (fans) and passive (doors and regulators). Today both can be controlled easily and cost effectively. Fans can be fitted with remote soft starters and their delivery controlled through either variable pitch or speed. Most commercial ventilation doors can be adapted for remote operation to turn the airflow on or off in specific areas. Some doors can also act as regulators when partially open; alternatively specific regulators with a finer control of volume may be required.

3. Compliance Monitors – to ensure the system is working effectively, two types of monitoring may be required. First, the airflow designed to go to an area must be guaranteed. For this non-intrusive ultrasonic airflow sensors are best suited to active mine airways however other technologies may also be considered in ducts. Second, the quality of the air must be ensured. For this there are a variety of single and multiple gas monitors available to measure the by-products of diesel activity. However, the placement of gas sensors and flow monitors can not always guarantee that an equipment operator is in “clean” air, so such systems may have to be supplemented with on-board gas monitors.

4. Data Management and Process Control System – this is the backbone and brains of the system by which information is gathered, commands sent and the outcome determined. The availability of reliable and cost-effective communications and programmable control has historically been another hurdle to the implementation of the demand concept. However, with the advancements of the information age, this demand concept has become more achievable.

![Figure 9 - Ventilation-On-Demand Based System](image-url)
5. ENVIRONMENTAL BENEFITS AS A RESULT OF REPLACING DIESELS WITH FUELCELLS

5.1 Greenhouse Gas Emissions

Canadian energy supply and demand and, thus, greenhouse gas emissions, are strongly influenced by “external” factors such as macroeconomic and demographic trends, international energy prices. The energy and emissions projections presented in Canada’s Energy Outlook (CEO97) were predicted on expected economic conditions from the perspective of late 1994. However, unforeseen events such as the Asian crisis, stronger economic growths in the United States, and better-than-anticipated fiscal conditions have altered the economic growth profile significantly over the short and medium-terms [20].

According to the Canada’s Energy outlook document the industrial sector, which includes all manufacturing industries, forestry, mining and construction (but excludes the oil and gas sector) is the largest energy-using sector. In 1997, the industrial sector produced 127 Mt of equivalent CO$_2$ emissions in greenhouse gas. Two-thirds of these emissions were from the consumption of energy and the remaining third from various industrial processes such as CO$_2$ release from cement production and anode production in aluminum smelting, adipic acid in chemicals and sulphur hexafluoride use in magnesium smelting. The six energy intensive industries such as chemicals, petroleum refining, mining and smelting, iron and steel, pulp and paper, and cement account for over 80% of the emissions [20].

Emissions from electricity generation alone in 1997 were 111 Mt of equivalent CO$_2$ emissions in GHG. The overwhelming proportion, 84%, was from the use of coal. Natural gas and oil accounted for 9% and 7% respectively. However, the restructuring of the electricity market in Canada is fundamentally altering the electricity supply industry. The basic trends include unbundling of major utility functions into generation, transmission and distribution. Restructuring should allow open access to transmission networks, thereby creating a more competitive generation market.

Figure 10, provides the overall projected trend in Canada’s greenhouse gasses from 1990 to 2020. It also offers an estimate of the magnitude of the Kyoto challenge expressed as the “gap” between the policy-as-usual and Canada’s target under the Kyoto’s Protocol, namely 6% below 1990 emission levels, on average, by 2010 and 8% below 1990 emission levels by 2015.

In 1990, Canada’s greenhouse gas emissions were 601 Mt of equivalent CO$_2$. By 1997, the latest year for which data are available, GHG emissions had risen to 682 Mt, a growth of 13%. The updated forecast suggests that, by 2010, Canada’s GHG emissions will increase to 764 Mt, and by 2020, to 845 Mt. It can be seen that by 2010, therefore, GHG emissions would be some 26% above the Kyoto protocol emissions level and by 2020, in the absence of GHG policy changes, they would be 34% above the Kyoto protocol emissions level (i.e. 845 Mt versus 553 Mt).
It should also be mentioned that this update incorporates estimates of the impact of the federal, provincial and municipal initiatives including the Voluntary Challenge and Registry program resulting from the 1993 national Action Program on Climate Change (NAPCC). Overall the NAPCC initiatives are estimated to reduce emissions by 60 Mt of equivalent CO$_2$ in 2010 and approximately 100 Mt of equivalent CO$_2$ in 2020. Had it not been for these initiatives, GHG emissions would have been almost 8% higher in 2010 and 11% higher in 2020. More specifically, in absence of these initiatives, the Kyoto gap would have been about 30% larger [20].

With regard to GHG emissions by sector, the fossil fuel industry is the largest contributor to emissions growth, with increases of approximately 64% between 1990 and 2010. This largely reflects the increase in oil sands production anticipated to occur during this period. The increase in GHG emissions from the industrial sector is also significant but the pace is somewhat slower, primarily due to greater energy efficiency improvements and reductions in certain process emissions (i.e. SF$_6$ from magnesium smelting). For electricity generation, emissions grow rapidly from 1990 to 2015. After 2015, however, growth declines sharply as existing coal-fired plants, reaching the end of their service life are retired and replaced by natural gas or highly-efficient coal-fired plants.

The industrial sector by definition is extraordinarily complex and heterogeneous. It includes all manufacturing, as well as metal mining, and construction activities. Within manufacturing, industries range from those that transform raw materials into more refined forms (e.g. the
primary metals and petroleum refining industries) to those that produce highly finished products (e.g. electronic industries). The most significant determinant of industrial emissions is demand for final outputs.

Table 5 - GHG Emissions in CO₂ Equivalent (Mt) by Industry Sub-Sectors

<table>
<thead>
<tr>
<th>Industry Sub-Sectors</th>
<th>1990 (Mt)</th>
<th>2000 (Mt)</th>
<th>2010 (Mt)</th>
<th>2020 (Mt)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pulp and Paper</td>
<td>12.6</td>
<td>11.3</td>
<td>13.3</td>
<td>13.7</td>
</tr>
<tr>
<td>Chemicals</td>
<td>26.8</td>
<td>20.2</td>
<td>23.0</td>
<td>26.9</td>
</tr>
<tr>
<td>Iron and Steel</td>
<td>14.1</td>
<td>16.0</td>
<td>16.2</td>
<td>17.5</td>
</tr>
<tr>
<td>Smelting and Refining</td>
<td>14.1</td>
<td>12.8</td>
<td>12.6</td>
<td>13.2</td>
</tr>
<tr>
<td>Petroleum Refining</td>
<td>17.8</td>
<td>21.4</td>
<td>23.3</td>
<td>26.9</td>
</tr>
<tr>
<td>Other Manufacturing</td>
<td>24.3</td>
<td>24.6</td>
<td>27.2</td>
<td>29.3</td>
</tr>
<tr>
<td>Metal Mining</td>
<td>4.7</td>
<td>6.9</td>
<td>7.7</td>
<td>8.5</td>
</tr>
<tr>
<td>Construction</td>
<td>0.7</td>
<td>1.3</td>
<td>1.2</td>
<td>1.2</td>
</tr>
<tr>
<td>Cement</td>
<td>9.7</td>
<td>10.0</td>
<td>12.0</td>
<td>14.7</td>
</tr>
<tr>
<td>Forestry</td>
<td>1.0</td>
<td>0.7</td>
<td>0.7</td>
<td>0.7</td>
</tr>
<tr>
<td><strong>TOTAL</strong></td>
<td><strong>125</strong></td>
<td><strong>125</strong></td>
<td><strong>138</strong></td>
<td><strong>152</strong></td>
</tr>
</tbody>
</table>

Table 5 shows the emissions anticipated from the industrial sector from various industries and groupings. Over the 2000-2020 period, total industrial emissions are projected to grow from 125 Mt to 152 Mt. This average growth is relatively slow (about 1% per year), and is generally due to a shift to less energy intensive industries and efficiency gains. While the emissions increase for the sector is approximately 10%, not all industrial sub-sectors are experiencing increases. Among the industry sub-sectors, the largest increases compared to 1990 emissions level are projected for construction (71%), metal mining (64%), petroleum refining (31%) and cement (24%). The largest declines in emissions are anticipated for forestry (30%), smelting and refining (11%) and chemicals (14%) [20].

Electricity use is currently the third largest source of GHG emissions. Although, the consumption of electricity produces no emissions at the point of use, its generation currently accounts for 15% of the total emissions, and that share is expected to increase to 16% by 2010. By 2020 the majority of the existing coal plants will reach their in-service life and will be retired, therefore, emissions from electricity generation could dramatically decrease.

Table 6 provides the projections for emissions from electricity generation by fuel type. This table shows that between 1990-2010, emissions from electricity generation are expected to increase by 25%. This table also shows several insights into the source of this growth. For example, emissions from natural gas-fired generation will dramatically increase by some 650%. This reflects the increasing importance of natural gas fired capacities. Between 1990-2010, emissions from coal-fired increase by 5%, while those from oil-fired generation are declining by as much as 55%.
Table 6 - GHG Emissions from Electricity Generation (Megatonnes)

<table>
<thead>
<tr>
<th>Electricity by Fuel Type</th>
<th>1990</th>
<th>2000</th>
<th>2010</th>
<th>2020</th>
<th>% Change Between 1990-2010</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal</td>
<td>80</td>
<td>90</td>
<td>84</td>
<td>49</td>
<td>5.0</td>
</tr>
<tr>
<td>Natural gas</td>
<td>4</td>
<td>18</td>
<td>30</td>
<td>69</td>
<td>650</td>
</tr>
<tr>
<td>Oil</td>
<td>11</td>
<td>3</td>
<td>5</td>
<td>5</td>
<td>-54.5</td>
</tr>
<tr>
<td>TOTAL</td>
<td>95</td>
<td>111</td>
<td>119</td>
<td>123</td>
<td>25.3</td>
</tr>
</tbody>
</table>

The more dramatic changes are expected to occur post-2010. Over the 2010-2020 period, the coal-fired related emissions decline from 84 Mt in 2010 to some 49 Mt in 2020. Offsetting this decline is a more than doubling of emissions related to natural gas fired generation. Natural gas fired emissions increase from 2010 level of 30 Mt to some 70 Mt in 2020. Oil fired generation is expected to remain the same, except in Newfoundland and in certain isolated communities. It should be noted that over the period of 2010-2020 the majority of Canada’s electricity production (i.e. over 75%) will be generated by non-emitting GHG sources: principally hydro and nuclear power [20].

5.2 Environmental Benefits From Using Fuelcell Powered Equipment

Table 5 (from Section 5.1 – Greenhouse Gas Emissions) shows that mine’s through their use of electricity and fossil fuels for intake airflow heating in the winter season and diesel powered vehicles will be responsible for generating approximately 7.7 Mt of GHG emissions by 2010. On the other hand, Figure 10 also shows that according to the Kyoto Protocol the predicted GHG emissions by 2010 would need to be reduced by as much as 26%. Consequently the Canadian industrial sector including the mining industry would need to find viable solutions in order to reduce their GHG emissions without compromising throughput and production rates. In regard to this the replacement of diesels with fuelcells can be extremely beneficial in reducing ventilation requirements and consequently the power consumption underground.

For all surveyed underground operations (See the Environmental Benefit section of the individually described operations – Appendix 1), a comparison in GHG emissions expressed in tonnes of equivalent CO₂ for the current situation (using diesel powered equipment), versus fuelcells is given in a table format. These comparisons show the reductions in electricity, natural gas and diesel fuel usage for fuelcells versus diesels and as a consequence the reductions in GHG emissions expressed tonnes of equivalent CO₂ per year. In order to express the GHG emissions for both current practice (using diesels) and fuelcells conversion factors such as 200 tonnes of equivalent CO₂ emissions per one giga watt hour (GWh) of consumed electricity (this is specific to Canada and reflects the make-up of the country’s electrical generation), 1.90 tonnes of
equivalent CO₂ emissions per one million litres (ML) of consumed natural gas, and 2.777 tonnes of equivalent CO₂ emissions per one million litres (ML) of consumed diesel fuel were used.

For all the surveyed mines, the total GHG emissions for the current practice and fuelcell option scenarios and their potential reductions are given in Table 7. Figure 11 shows the GHG emissions and their potential reductions in a graphical format.

### Table 7 - GHG Emissions and Their Potential Reductions Fuelcells vs. Diesels

<table>
<thead>
<tr>
<th>Underground Operations</th>
<th>Total Greenhouse Gas Emissions</th>
<th>Potential Reductions</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Using Diesels - In equivalent CO₂ (Tonnes/year)</td>
<td>Using Fuelcells - In equivalent CO₂ (Tonnes/year)</td>
</tr>
<tr>
<td>Holloway Mine:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Electricity</td>
<td>1179.0</td>
<td>818.0</td>
</tr>
<tr>
<td>Propane/Natural Gas</td>
<td>2.8</td>
<td>2.2</td>
</tr>
<tr>
<td>Diesel Fuel</td>
<td>2.2</td>
<td>0.8</td>
</tr>
<tr>
<td>Total Holloway</td>
<td><strong>1,184.0</strong></td>
<td><strong>821.0</strong></td>
</tr>
<tr>
<td>Creighton Mine-Upper OB:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Electricity</td>
<td>17,818.0</td>
<td>11,054.0</td>
</tr>
<tr>
<td>Propane/Natural Gas</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Diesel Fuel</td>
<td>3.0</td>
<td>1.0</td>
</tr>
<tr>
<td>Total Creighton-Upper OB</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td><strong>17,821.0</strong></td>
<td><strong>11,055.0</strong></td>
</tr>
<tr>
<td>Creighton Mine-Deep OB:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Electricity</td>
<td>21,818.0</td>
<td>17,580</td>
</tr>
<tr>
<td>Propane/Natural Gas</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Diesel Fuel</td>
<td>3.0</td>
<td>1.0</td>
</tr>
<tr>
<td>Total Creighton-Deep OB</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td><strong>21,821.0</strong></td>
<td><strong>17,581.0</strong></td>
</tr>
<tr>
<td>McCreedy East:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Electricity</td>
<td>15,231.0</td>
<td>9,703.0</td>
</tr>
<tr>
<td>Propane/Natural Gas</td>
<td>4.8</td>
<td>3.8</td>
</tr>
<tr>
<td>Diesel Fuel</td>
<td>1.5</td>
<td>0.4</td>
</tr>
<tr>
<td>Total McCreedy East</td>
<td><strong>15,237.0</strong></td>
<td><strong>9,707.0</strong></td>
</tr>
<tr>
<td>Laronde Mine:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Electricity</td>
<td>30,368</td>
<td>21,625.0</td>
</tr>
<tr>
<td>Propane/Natural Gas</td>
<td>9.6</td>
<td>8.3</td>
</tr>
<tr>
<td>Diesel Fuel</td>
<td>5.9</td>
<td>1.5</td>
</tr>
<tr>
<td>Total Laronde</td>
<td><strong>30,384.0</strong></td>
<td><strong>21,635.0</strong></td>
</tr>
<tr>
<td>Louvicourt Mine:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Electricity</td>
<td>4,645.0</td>
<td>3,179.0</td>
</tr>
<tr>
<td>Propane/Natural Gas</td>
<td>13.8</td>
<td>11.3</td>
</tr>
<tr>
<td>Diesel Fuel</td>
<td>1.5</td>
<td>0.4</td>
</tr>
<tr>
<td>Total Louvicourt</td>
<td><strong>4,660.0</strong></td>
<td><strong>3,191.0</strong></td>
</tr>
<tr>
<td>Doyon Mine:</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Electricity</td>
<td>10,402.0</td>
<td>7,572.0</td>
</tr>
<tr>
<td>Propane/Natural Gas</td>
<td>2.5</td>
<td>2.0</td>
</tr>
<tr>
<td>Diesel Fuel</td>
<td>3.1</td>
<td>0.8</td>
</tr>
<tr>
<td>Total Doyon</td>
<td><strong>10,408.0</strong></td>
<td><strong>7,575.0</strong></td>
</tr>
<tr>
<td>Total GHG Reductions</td>
<td><strong>101,515.0</strong></td>
<td><strong>71,565</strong></td>
</tr>
</tbody>
</table>
Table 7 shows that for the surveyed Canadian underground metal mines, the average potential GHG emissions reductions per operation is approximately 4,280 tonnes/year of equivalent CO₂ or a 29.5% reduction. If this value could be extrapolated across Canada, the total GHG emissions reduction as a result of replacing diesels with fuelcells would be around 420,000 tonnes of equivalent CO₂/year.

Looking at the overall projected trend in Canada’s GHG emissions from 1990-2015 it can be observed that the Kyoto gap by the year 2003 can be at approximately 75 million tonnes of equivalent CO₂ or approximately 10% above the levels stipulated in the Kyoto Protocol. Table 5 shows the predicted GHG emissions from the industrial sector including the metal mining industry over the period of 1990-2020. According to this, the metal mining industry’s GHG emissions by 2003 are evaluated at 7.1 million tonnes equivalent CO₂ per year. In order to address the Kyoto Protocol’s gap, by the end of 2003 the metal mining industry’s GHG emissions would need to be reduced by approximately 10%, in other words by an equivalent of 710,000 tonnes of equivalent CO₂.

It can be concluded that if the primary diesel powered underground equipment would be replaced by fuelcells, reductions in GHG emissions from the primary and auxiliary ventilation systems alone across Canada would be in the range of 420,000 tonnes of equivalent CO₂ emissions per
year, which would represent approximately 59% of the total GHG emissions reductions required by the Kyoto protocol.

It can be seen that apart from capital and operating cost reductions, health and safety benefits, the fuelcells technology could have a significant impact with regard to the global environment.

6. **MINE VENTILATION QUALIFIERS THAT MIGHT LIMIT COST BENEFITS**

The most important qualifiers found to limit capital and operating cost savings in precious and base metal mines are as follows:

1. In the preceding analysis, one of the potential qualifiers to the introduction of fuelcells was considered, namely drift velocities. In standard ventilation texts recommended air velocities range from 1 m/s in cool environments through to 3 m/s in hot environments. Within most of the Canadian underground operations where heat is not a problem low airflow velocities are acceptable. For some of the underground operations within the fuelcell option analysis lower velocities have been considered, but limited to more than 0.5 m/s. This is because below this airflow velocity, air movement is barely perceptible and problematic to measure.

2. Stope velocities were another concern at some of the underground operations mostly because of blast fume clearance issues, which under the current ventilation regimes required up to 2 hours, especially in case of long hole blasts. To avoid prolonging this time to the detriment of production, the mines were required to retain high volume flush capacities within their secondary auxiliary ventilation system. As a result the air volumes within the auxiliary ventilation system could not be trimmed at the same level as the air volumes within the primary ventilation system. As a consequence, power and operating cost reductions within the auxiliary systems were much less than those experienced within the primary ventilation system (see Figures 4,5).

3. Shaft velocities are potentially another issue. It is normally recommended that air velocities between 7 & 12 m/s be avoided in exhaust shafts or raises. This is the critical velocity range in which water droplets can become suspended and cause adverse affects on primary fan systems. As a result in order to maintain exhaust airflow velocities in the exhaust shafts above the 12 m/s velocity range, the air volumes in some active areas have not been trimmed, despite reduced airflow needs by the fuelcell powered equipment. Within the fuelcell scenarios this had a negative impact on both primary and auxiliary ventilation operating costs.

4. The air velocity across a mine’s heating system is also a concern especially if the primary flow is supplied by a single fan. At most operations, the surface fan installation comprised two fans in parallel, and the ventilation simulation exercises with reduced flow indicated that a single fan was able to provide the required airflow. In situations where two fans were still needed to deliver the required intake air volume and different blade settings or fan motors RPM reductions were used to accommodate reduced intake airflow
regimes, the airflow velocities across the burners were still sufficient in order to avoid negative impact upon the heating system.

5. Another issue at some operations was silica exposure. For example at the Holloway mine, within the current ventilation scenario with airflow provided based upon diesel exhaust, the mine wide silica exposure was 0.06 mg/m³. Consequently, with ventilation reduced by a maximum of 24% and if applied uniformly across the mine, the dust concentration could increase to 0.08 mg/m³. This is still below the Ontario exposure limit for $\alpha$-quartz of 0.10 mg/m³, but there could be a higher incidence of individual values approaching or even exceeding this limit.

In addition to the mitigating qualifiers already addressed, in the base and precious metal mines there are other issues yet to be resolved, such as:

- In most environments, a fuelcell is considered a zero emission power unit. In the mining atmosphere the obvious elimination of diesel exhaust and its potentially carcinogenic constituents are beneficial. However, fuelcells will generate heat and water in their exhaust. In terms of heat, the deep metal mine analysis has shown that a fuelcell could require less air to remove heat than a comparable diesel, until heat from other sources (i.e. strata) becomes the over-riding issue. With respect to the addition of moisture to the environment, this could be problematic in certain mines where the exhaust air cools as it ascends access ramp systems as fogging may result. However, at this time it is not possible to state whether the situation would improve or worsen under a reduced flow regime until the true moisture contribution of fuelcells after in-exhaust condensation has been established.

- Another issue is the amount of ventilation required to dilute a possible hydrogen leak from the fuelcell. This is another potentially limiting factor that would need to be evaluated.

- It should also be remembered that for three underground operations the fuelcell benefit analyses were performed according to a design criteria that came under Ontario’s Provincial jurisdiction where airflow requirements are based upon 0.063 m³/s/kW diesel power. In other jurisdictions such as the province of Quebec, diesel airflow requirements acknowledge the use of clean diesel fuel and “clean” diesel engine technology. In Quebec, a primary production diesel unit could require up to 40% less air than in Ontario. Consequently, if all provinces accepted the use of “clean” diesel engine technology and it was adopted throughout mine’s diesel fleets; their current airflow requirements would be reduced and as result the primary and auxiliary ventilation benefits would have been somehow less significant.
7. CONCLUSIONS

Fuelcells have the potential to dramatically reduce the air volumes required in underground mines, however sufficient ventilation must be retained to control dust and blast fume clearance plus to support the remaining light-duty diesel equipment. They also appear to significantly reduce mines GHG emissions, therefore addressing increasing global environmental concerns.

The preceding analyses show potential for 24 – 53% reductions in the electrical operating cost of the primary ventilation system for associated average flow reductions of 9% (with workings located below 2450 m depth) and 24% with workings located at relatively shallow depth (between 400 – 800 m depth).

Reductions in the electrical operating costs of the production auxiliary fans are only in the order of 11% to 20%, irrespective to the change in surface fan flow. These savings are much less than those experienced within the primary ventilation system mostly due to reduced flexibility of the auxiliary ventilation system (lack of variable speed drives and ventilation controls), to accommodate reduced airflow regimes. As mentioned before, unless the mining operations are willing to replace the auxiliary fans to accommodate new airflows, reduced airflows could only be obtained by throttling the current auxiliary systems at their discharge, a practice which would not change the power consumption and consequently the auxiliary system’s operating cost.

Further to this, in captive underground areas, with operations employing a “series” auxiliary ventilation practice, any reduction in airflow as a result of replacing diesels with fuelcells could increase the blast fume clearance time and increase dust concentrations within the working areas. Therefore, the mines would need to retain a high volume flush capacity. As a result, the air volumes and consequently the operating costs could not be reduced at the same level as within the primary ventilation system.

Heating cost reductions as a result of replacing diesels with fuelcells were the same as the average mine intake airflow reductions (i.e. between 9 – 24%). After factoring in the relative magnitude of each cost (i.e. primary, auxiliary and heating systems), the combined savings in electrical power and heating were between 20% (Creighton Mine – Deep Orebody) and 38% with workings at shallow/medium depth.

Canadian mines through their use of electricity, diesel fuel for the diesel-powered underground equipment and fossil fuels for heating could be responsible for generating approximately 7.7 million tonnes of CO₂ equivalent greenhouse gas emissions by 2010. On the other hand, according to the Kyoto Protocol, which has been ratified by Canada, the predicted total GHG emissions across Canada would need to be reduced by as much as 26% (See Figure 10 – Kyoto gap). Consequently, the Canadian industrial sector, which includes the metal mining sector, would have to identify viable solutions in order to reduce their GHG emissions.

As a result of replacing the primary diesel powered equipment with fuelcells, these analyses show reductions in GHG emissions between 27 to 38%, or from 363 tonnes of CO₂ equivalent (at the Holloway mine) to 8,749 tonnes of CO₂ equivalent (at the Laronde mine) per year. The metal mining industry’s GHG emissions by 2003 are estimated at 7.1 million tonnes of CO₂ equivalent per year. According to the Kyoto Protocol, in 2003 the metal mining industry would
need to reduce its GHG emissions by approximately 10%, or an equivalent of 710,000 tonnes of CO₂. If the primary diesel powered underground equipment would be replaced by fuelcells, reductions in GHG emissions from the mines ventilation system alone across Canada would be in the range of 500,000 tonnes of CO₂ equivalent per year. This represents about 70% of the metal mining industry’s required GHG emissions reduction as per the Kyoto protocol. So, it can be seen that the fuelcell technology could provide a viable technical solution in mitigating the increasing global environmental concerns.

With respect to relative economics, the area where fuelcells are most beneficial is in reducing the operating cost of the primary ventilation system that provides the bulk flow through the mine and the GHG emissions. The next best area is in mine heating, which is linearly related to the mine’s total intake airflow. The area least affected is in auxiliary fan power consumption.

As to how these relative savings combine at any mine is dependant upon the proportion of the mine’s cost spent for primary fan electricity, auxiliary fan electricity and heating fossil fuels. In one shallow operation (i.e. Holloway mine), despite significant decreases in electrical power consumption by the primary fan system, propane-heating costs was the overriding element. In deep operations (i.e. Creighton mine – deep orebody), the primary system electrical costs were the most significant but the overall benefit was reduced through near stable auxiliary costs. At the Holloway mine, analysis also showed a reduction in potential cost-benefit as the mine was already varying its intake airflow according to production needs. As a result of this mine being inactive 33% of the time, the relative savings across the remainder (although significant) were reduced in the overall analysis.

Even in those mines where the ventilation economic advantage is relatively small, it must be remembered that fuelcells with zero emissions will be extremely beneficial to the health and safety of the underground workforce through the elimination of diesel contaminants. Economical, workforce health and safety benefits and their potential to address increasing global environmental concerns can make the fuelcell technology an extremely attractive alternative within the mining industry.

8. FUTURE WORK

So far, this analysis has shown that fuelcells could be extremely beneficial in mines where production is continuous and heat is not an issue. However, heat and existing ventilation management issues are qualifiers to the potential benefits. Similarly, the need to maintain minimum airflow velocities within the working areas (and even the inactive areas) can limit environmental and cost benefits. In addition there may be other mine specific qualifiers, such as inherent dust conditions, shaft and heating system velocity limits, which are characteristic to certain underground operations.

As a result, it is due to the relevance of such qualifiers in Canada that mines have to be evaluated on an individual basis and the analyses presented in this report may not truly reflect the impact of the whole metal mining industry. For a more accurate impact analysis of the fuelcell technology the majority of Canadian underground mines would need to be analyzed on an individual basis.
For shallow and medium depth mines where heat wasn’t an issue, the airflow requirements for the primary fuel cell powered equipment have been determined based upon the mine’s production rate such as 0.042 m³/s per tons/day (90 cfm/tpd) plus an additional 25% air volume allowance for miscellaneous needs. The mine’s total intake airflow was then revised according to the airflow distribution and minimum airflow velocity requirements.

For deep underground operations, to obtain the intake airflow needs for fuelcells, the relative equipment and non-equipment heat components at varying depth was determined from the mine’s production flow requirements and knowledge of rock thermal properties by means of an underground climatic simulator. However, a portion of the heat generated by the fuelcells is used to release the hydrogen from its hydride bed, and as a result less heat is transferred into the ventilation air. On the other hand, fuelcells have the potential to generate more moisture than equivalent diesels units, which combined with reduced air volumes could lead to more humid conditions in mines. Therefore, for a more accurate evaluation of the intake airflow needs, which ultimately will affect the mine’s total intake airflow and consequently the cost benefits, a fuelcell-powered LHD “in-situ” underground testing program is required.

9. ACKNOWLEDGEMENTS

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APPENDIX 1  -  VENTILATION COST BENEFIT ANALYSIS  
(DETAIL DESCRIPTION)

A.  HOLLOWAY MINE - BATTLE MOUNTAIN CANADA

1.  INTRODUCTION

An underground gold mining operation in northern Ontario, Canada, has been used as a case-study for the potential ventilation related benefits of introducing fuel-cell based technology to their primary production diesel powered fleet of mobile vehicles. The mine produces 2,300-2500 tpd, has 132 employees, with an operation schedule of three 8-hr shifts per day, 5 days a week.

The orebody is 285-795 m below surface, well defined and is now extracted through a longitudinal retreat stoping method. Typical stope dimensions are 8 m wide, 15 m high and 25 m long. Typical drift development is 4m wide by 3.7 m high.

The mine is heavily dieselised utilizing, diesel powered equipment in all three phases of mining: drilling, mucking and haulage. Drilling is performed with 61 kW (82 hp) diesel-electric drill-jumbo’s powered by Deutz F6L912W engines. Mucking is performed with 186 kW (250 hp) 7 yd³ bucket scooptrams powered by Detroit Diesel Series 50 engines. Material is hauled in combination by the scooptrams and 16 ton Trucks powered by 138 kW (185 hp) powered by Deutz F8413FW engines. For ground control the mine also uses cemented and plain rockfill, which is also placed by the mine’s “production” scoops at the rate of 400 tpd.

The basic layout of the mine is shown in the ventilation schematic, Figure 1. The mine has a single primary intake raise down to the 400 m Level fed by two surface parallel Joy M84-50-1180 fans with variable speed drives and 375 kW (500 hp) motors. This raise then connects to an old internal raise system to take the air down to the 525 m at which point the air crosses the mine to another internal raise system that takes the air down to the crusher at 770 m. Air to production areas, is drawn off of this intake system either naturally or under the assistance of auxiliary fans. Auxiliary ducting then takes this air to the desired production locations. In the production areas, single-pass ventilation is employed. Air is then exhausted: from the bottom of the mine through an exhaust raise system between 650 m Level and the 485 m Level, or through either ramp system to then leave the mine via the Exploration or Production shafts. The raise/ramps/shafts are open on several levels to permit the exhaust flows to balance themselves for minimum resistance. If not containing material the backfill raises are also available as exhaust routes.

2.  CURRENT VENTILATION REQUIREMENTS AND DESIGN CRITERIA

In the current system, the controlling criteria for ventilation is diesel exhaust dilution, secondary considerations are blast fume clearance and dust control. Due to its location, northern Ontario, and the relatively shallow depth of the workings <800 m, heat is not currently an issue. Due to
the ore/strata of the mine there are no other strata gas issues such as methane that form part design criteria.

Table 1 lists the mine’s underground diesel equipment inventory, their fleet total is 34 operating vehicles with a combined 3,043 kW (4,082 bhp), with individual engine sizes ranging from 187 kW (250 bhp) scooptrams down to a 37 kW (50 bhp) utility vehicle. Based upon this equipment list and the applicable legislation in Ontario of 0.063 m³/s/kW (100 cfm/bhp), the

Figure 1 - Ventilation Schematic – Holloway Mine
mine should be supplying 192.4 m$^3$/s (408 kcfm) if all the equipment were operating, however after subtracting typical vehicles under maintenance the requirement drops to 180 m$^3$/s (381 kcfm) for the two productions & development shifts each day. On the third daily production shift (nights), both fans are turned back from 95% to 85% of rated speed. On weekends and statutory holidays, the mine reverts to a single fan operating at 65% of rated speed.

Using a 3D-CANVENT model of the mine’s ventilation system, updated September 1999, to deliver 180 m$^3$/s, the mine’s primary intake fans would be developing 2.45 kPa (9.8"wg), for a combined air power of 441 kW and running at 82% efficiency. The mine would be operating at this level of ventilation 45% of the time and with electricity charges of $0.067(Can)/kWh, its annual operating cost would be $143k(Can). At 85% of the rated fan speed the flow drops back to 160.5 m$^3$/s at 1.97 kPa (7.9"wg), 316 kW of air power and retains the same order efficiency. The mine would operate at this level of ventilation 22% of the time with an annual operating cost of $50k(Can). During non-active periods when the mine reverts to a single fan at 65% of the rated speed, the mine’s airflow drops to a flow of 82.0 m$^3$/s at 0.51 kPa (2.0 "wg), 42 kW of air power and the fan efficiency drops to the order of 60%. The mine would be operating at this level of ventilation 33% of the time with a resultant annual cost of $14k(Can). Based upon these three levels of ventilation the combined annual operating costs of the surface fan would be $207k(Can) for an average flow of 143.0 m$^3$/s.

For the secondary ventilation distribution to production areas, the mine has 30 auxiliary systems with fan powers ranging from 22.3 kW (30 hp) through to 112 kW (2 x75 hp). The combined installed power of the mine’s 31 fans is 1,650 kW (2,211 hp). However, these fans would only be operational according to need. Consequently, only 33%, sufficient for 10 active locations, may be operating during the mine’s 2 daily production/development shifts, and only 20%, sufficient for 6 active locations, may be operating during the production night shift. During the weekend and statutory holidays all auxiliary systems should be turned off. Assuming these fans are running at their rated power and using a $0.0675(Can)/kWh electricity charge, their operating cost would be of the order of $145k(Can) during the production/development shifts, and $43k(Can) during the production night shift. Based upon these assumptions the combined annual operating costs of the production auxiliary fans would be $188k(Can).

In combination, the primary and production orientated auxiliary fans would cost $395k(Can) per year, which reflects a consumption of 5.9 GWh. This cost represents approximately 23% of the mine’s total electricity charges.

In addition to this, in 2002, the mine also spent another $576k(Can) for 1.8M litres of propane to heat their air during the winter season.

Combining electrical power and heating, the ventilation operating costs of this mine are currently $971k(Can).
3. FUTURE VENTILATION REQUIREMENTS USING FUELCELLS

Determining the appropriate ventilation requirement in a mine employing fuel cells presents some difficulty depending upon the degree of conversion and how the other diesel vehicles are accounted for in the original design volume.

1. Based upon production rate and using the 90 cfm/tpd plus 25% additional allowance. Assuming the higher production rate of 2500 tpd the mine would require 132.5 m$^3$/s (281 kcfm), a 26% reduction. To deliver this flow, 3D-CANVENT simulations show that the mine would probably only require one of its two surface fans to operate but at a higher blade angle, to develop 1.36 kPa (5.5 ”wg) and its efficiency could drop off to at best 70%. The resultant annual operating cost for the production/development shifts drops to $68k(Can), a 52% reduction.

As this reduced volume is based upon production, which continues through all three shifts, the volume supplied should remain the same during the night shift. Consequently, the resultant annual cost for the production only shift would be half that of the production/development shifts, i.e. $34k(Can), a 32% reduction.

For the weekend and statutory holidays, the magnitude of their reduced volume and consequent costs would not change. Therefore, the overall annual costs of the primary ventilation system based upon production rate would be $116k(Can), a 44% reduction, for an average flow of 116.0 m$^3$/s (246 kcfm), a 19% reduction.

For this reduced volume, by proportion, the annual propane heating cost would reduce to $467k(Can), the same 19% reduction. Combined these represent a 26% saving in costs for the primary ventilation system.

Operating cost savings in the secondary ventilation system are harder to estimate because unless the mine changes the auxiliary fans, reduced flows would be obtained by throttling the current systems at their discharge. This is the mine’s current practice for reduced flow requirement operations in a high flow system. This practice would not change the operating costs of the secondary system.

However, if the mine was to down-size their current “production” operation fans to the next lowest size in their inventory, i.e.100 hp to 75 hp, and 75 hp to 64 hp, their total fan power would be reduced by 12% and a similar reduction in operating costs to $166k(Can), would follow.

In combination, the primary and production orientated auxiliary fans would cost $282k(Can) per year, which reflects a consumption of 4.2 MWh, a 29% reduction.

Combining electrical power and heating, the ventilation operating costs of this mine based upon the 90cfm/tonne plus a 25% allowance would be $749k(Can), a 23% reduction.

2. Based upon remaining diesel-powered equipment. Assuming fuel-cells will only be replacing the primary vehicles, i.e. the 7.5 yd$^3$ production and 3.5 yd$^3$ development
scooptrams and the 16tonne trucks, from Table 1, it can be calculated that the mine’s remaining diesel requirement would be reduced to 98 m$^3$/s (208 kcfm), a 46% reduction. If this air is sufficient for both the diesel activity and blast/dust concerns modelling shows that this volume could be delivered at 0.73 kPa (2.9 “wg), by a single fan operating at best with a 65% efficiency. The resultant annual electricity operating cost would be $44k(Can) for all three of the production shifts and remain at $14k(Can) for the non-productive periods. Under this regime the operating costs of the primary ventilation system becomes $58k(Can), a 72% reduction, for an average flow of 93 m$^3$/s (197 kcfm), a 35% reduction.

Similarly heating costs would drop by 35% to $375k(Can). Combined these show a 45% reduction in primary ventilation costs.

With this design criterion, the requirements for auxiliary ventilation are even harder to define than under the previous production rate rationale. However, it can be still assumed that auxiliary ventilation systems will be designed for the diesel-powered support vehicles. Consequently, 64 hp fans may still be the norm and the resultant operating cost savings would be virtually the same as in the previous scenario, namely a 12% saving.

In combination, the primary and production orientated auxiliary fans would cost $224k(Can) per year, which reflects a consumption of 3.3 MWh, a 44% reduction.

Combining electrical power and heating, the ventilation operating costs of this mine based upon supplying air for the remaining diesels would be $599k(Can), a 38% reduction.

3. Evaluating ventilation need on an individual operation basis: Table 2 shows the breakdown of the ventilation distribution for this gold mine showing 9 primary active areas (3 mucking, 3 drilling & 3 development) and four other specific requirements. This table also includes the justification for each flow. In total, these 13 requirements account for 139 m$^3$/s (295 kcfm) during the production/development shifts, and another 41 m$^3$/s (87 kcfm) or 30% is used to ventilate other non-specific areas and take account of leakage. This results in a total flow of 180 m$^3$/s (381 kcfm). During the production nightshift, the flow requirement drops to 150 m$^3$/s (317 kcfm), if assuming the same 30% oversupply, this is less than the 160 m$^3$/s (339 kcfm) the mine actually supplies.

Table 3 shows the ventilation needed on the replacement of diesel power units on the scooptrams and trucks by fuel cells plus retaining a 30% additional volume for other requirements and leakage. In the first part of Table 3 the ventilation requirements have been based upon the diesel requirements of the next largest vehicle (a service unit). In this table it can be seen that the overall demand drops to 101 m$^3$/s (214 kcfm) during the production/development shifts. However, it is apparent that with this level of flow air velocities drop to 0.34 m/s (59 fpm). This is below what would probably be acceptable for manned working areas. This is re-addressed in the second part of Table 3, which shows a revised requirement of 137 m$^3$/s (290 kcfm) during the production/development shifts. Applying the same rationale to the production nightshift its requirement becomes 108 m$^3$/s (230 kcfm), and the average volume becomes 112 m$^3$/s (238 kcfm), this is comparable to the modeled solution in the first analysis.
The above analyses show potential for a 44-71% reduction in the electrical operating costs of the primary ventilation system for associated average flow reductions of 19% and 34%. Reduction in heating costs would mirror the flow reductions, at 19% and 34%. Reductions in the electrical operating costs of the production auxiliary fans would be of the order of 12% irrespective of the change in surface fan flow. After factoring in the relative magnitude of the costs of each, the combined savings in electrical power and heating for the mine become 23% and 38% for the respective flow reductions of 19% and 34%. These relative savings are much less than those experienced in the operating cost of the surface fan because this mine needs to heat its air, and the auxiliaries may not be trimmed to the same degree as the primary system. It should also be noted that some of the potential savings that could have materialized, have already been reduced through this mine employing three levels of ventilation that reflect “production” needs.

Regardless, both still show significant cost savings, but the viability of these new flows has to be established.

4. AIR VELOCITY CONSIDERATIONS

- Exhaust Shaft Velocities – As the mine has two well sized shafts (4.9 & 5.5 m diameter) the velocities in the shafts are well below critical velocities even before any reduction in flow was considered. Consequently, suspended water droplets would not be an issue.

- Burner Velocities – As the reduced flows can be handled by one fan, if their intakes are separate, reasonable velocities should be maintained across the propane burners. This is important in cold climates because low air velocities and possibly resultant air density differences can cause fan impeller failures.

- Drift Velocities – Recommended air velocities in drifts range from 1 m/s (200 fpm) through 2 m/s (400 fpm) to 3 m/s (600 fpm) for cool, typical and hot environments. It should also be remembered that velocities below 0.5 m/s (100 fpm) are barely perceptible and can become problematic to measure. Consequently at this mine, which may be termed cool (13-20 °C) for the most part, 0.5-1 m/s (50-100 fpm) may be the low velocity limit. Therefore, for the mine’s typical drift size of 3.7 m high by 4.0 m wide, the minimum flow should be >7.4 m³/s (16 kcfm) and possibly 14.8 m³/s (31 kcfm).

- Stope Velocities – For this mine the primary consideration is blast fume clearance, with the current level of ventilation, development blasts are cleared within 30 minutes and long-hole blasts can take in excess of 2 hours to clear. However, these are normally scheduled at the end of the production week for clearance across the non-active weekend. As this is a 3 shift per day production operation an increase in blast fume clearance times may not be acceptable. For the development areas, the mine currently allocates 8m³/s (17kcfm) for a 3.5 yd³ scooptram, however on the introduction of fuelcells the diesel requirement of a development drops to 5 m³/s for a service vehicle. As a consequence of this 37% reduction in flow, the clearance time would proportionally increase by 60% to 50 minutes, which may be unacceptable. To combat this the mine could use a high-
volume flush for blast clearance. However, due to low air velocity concerns, the flow in the development areas may only be reduced to 7.5 m$^3$/s (16 kcfm), which would have negligible impact on fume clearance times.

As there will be instances when long-hole blasts occur during the week, rather than allocate higher flows routinely, the mine should retain the flexibility of a high volume flush. This could even be greater than it is currently supplying, for assuming, an air velocity limit of 0.3 m/s (60 fpm) for blast clearance, stopes with typical dimensions of 15 m high by 8 m wide would require 36 m$^3$/s (76 kcfm).

5. OTHER CONSIDERATIONS

1. Dust Conditions – Currently, with ventilation provided at diesel prescribed levels, the mine has a mine wide average silica concentration of 0.06 mg/m$^3$. Consequently, if the ventilation is reduced the dust concentrations could increase. In Ontario, the exposure limit for α-quartz (the type of silica normally quantified in mine air samples), is 0.10 mg/m$^3$, therefore ventilation reductions may be limited to at most 40%. However, to avoid exposures at the limit a lower reduction in ventilation would have to be considered.

2. Fogging Issues – As is typical with many Ontario metal mines, their exhaust air reaches its saturation point as it ascends and cools. This is not normally an issue in exhaust raises unless suspended water droplets form. However, in mines with exhaust shafts it can severely hamper visibility, and thus impact on safety, in their headframes. At this mine, the potential for fogging in the head-frame is addressed with local heating.

Fogging within the working area of a mine is a more serious issue. In a multi-level mine such as this air ascending an extensive ramp system will naturally cool through decompression and mixing with colder airstreams. This is known to occur at Holloway in the summer when the air entering the mine is already relatively humid before picking up additional moisture within the mine.

In a mine with reduced air volumes and fuel cells replacing the diesel engines in the larger equipment the potential for fogging will increase as a consequence of:

a) With reduced air volumes entering the mine, the temperature of the air travelling to the workplaces will increase due to a larger strata heat component. This would permit a greater uptake of available moisture.

b) Fuelcells generate more water than their equivalent diesel counterpart, unless they have a water scrubber on their exhaust. A 186 kW (250 hp) fuel cell would generate (operating at its rated power) 0.0204 kg/s of water {Schnackenburg, 2002}. For the equivalent diesel with an electronically controlled engine, it can be estimated that it would produce 0.0105 kg/s after subtracting water contained in the intake air {Gangal, 2003}. 
c) If the fuel cell works in an airflow that is 50% of that supplied for the diesel engine, and due to the fuel cell producing approximately double the amount of water, the net increase in moisture added to a unit volume of air could increase four-fold. This would probably be sufficient to bring the exhaust air of the operation to its saturation point. Subsequently, any slight reduction in temperature would result in fogging.

As a result of the potential dust and fogging concerns, this mine may be limited to at best a 30-40% reduction in the maximum production/development shift flow within the mine. This reduced flow should probably be maintained across the production night shift, and the flows would remain unchanged across weekends and statutory holidays. Consequently, for a 30% reduction in the maximum flow, the mine’s average flow only drops 22%. Due to this potential limit, the viability of introducing fuel cells as opposed to other airflow reducing requirements for diesel engines should be considered.

6. DIESEL AIRFLOW REQUIREMENTS

As stated under the current ventilation requirements, the current design criterion in Ontario is 0.063 m³/s/kW (100 cfm/bhp) regardless of engine type, however in other jurisdictions CSA standard requirements that are specific to an engine and take account of “clean-engine” technology have been adopted. Consequently, the engine on this mine’s production scoop, a Detroit Diesel Series 50 would only require 0.044 m³/s/kW (70 cfm/bhp) which is 30% less. If the diesel engines on the rest of the mine’s scoops and trucks were replaced with cleaner alternatives, similar reductions in required flow would occur.

On repeating the ventilation requirements of the individual operations and assuming Ontario accepted the CSA requirements, Table 4, initially shows that the ventilation could be reduced by up to 27% during the production/development shifts. However some of the velocities in the drilling and development areas are less than 0.5 m/s (100 cfm). On instituting this second limiting criterion, the potential flow reduction during the production/development shifts is reduced to 19% as compared to 24% for the fuel cell with the same qualifier.

After factoring in the different shift flows, the net reduction in average flow for the clean diesel option is 16% as opposed to 20% for the fuel cell. Consequently, the annual operating cost of the ventilation system including heating for the “clean” diesel option would be $780k(Can), a reduction of 20%, and for the fuel-cell $731k(Can), a reduction of 25%.

The difference between the two options is only $50k(Can) from the ventilation economics standpoint, however fuel cells have other health and environmental benefits. Obviously, fuel cells do not create diesel soot (a suspected carcinogen), which is good for the workforce. However in the “absence” of diesels which have historically dictated high volumes of air that were sufficient for most other contaminants, with lower flows other potential health hazards such as silica can re-emerge.
7. ENVIRONMENTAL BENEFITS

Mine’s through their use of electricity and fossil fuels for heating and powering diesel vehicles are responsible for generating greenhouse gases (GHGs). Globally, there is a desire for these GHG emissions to be reduced. Consequently, fuel cells can be beneficial in mines through replacing diesels and reducing ventilation requirements.

In Table 5, a comparison is given of the equivalent CO₂ emissions (tonnes) for the current operation, on the adoption of fuel cells, or use of “clean” diesels based upon their respective ventilation requirements as previously derived. In the fuel-cell option the diesel consumption has been reduced to 25% for the remaining non-production small vehicles based upon mine supplied information. No reduction of diesel consumption has been included for the “cleaner” diesels. The conversion factors used were 950 tonnes/GWh of electricity generated (this is specific to Canada and reflects the make-up of the country’s electrical generation primarily from hydro-electric stations), 1.55 tonnes/ML propane, and 2.73 tonnes/ML diesel.

Based upon these assumptions it can be seen that the fuel cell option would reduce this mine’s attributable GHG emissions by 3.9 tonnes or 37%. In comparison, maintaining “clean” diesels would only reduce the mine’s emissions by 2.5 tonnes or 24%. However, these estimates do not include the upstream effects of producing the fuels used at the mine or for electricity generation.
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<tr>
<td>Scoop - 2.2yd³ (Fork-utility)</td>
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<td>75</td>
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<tr>
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<td>Deutz F6L912W</td>
<td>68</td>
<td>91</td>
<td>9.1</td>
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<tr>
<td>Service-Scissor Lift (Hy-Hab)</td>
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<td>Deutz F6L912W</td>
<td>68</td>
<td>91</td>
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<td>Service-Scissor Lift</td>
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<td>Deutz F6L912W</td>
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<td>Drill-Jumbo</td>
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<td>Deutz F6L912W</td>
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<td>82</td>
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<td>Drill-Jumbo</td>
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<td>Deutz F6L912W</td>
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<tr>
<td>Drill-Jumbo</td>
<td>103</td>
<td>Deutz F6L912W</td>
<td>61</td>
<td>82</td>
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<tr>
<td>Drill-Jumbo</td>
<td>104</td>
<td>Deutz F6L912W</td>
<td>61</td>
<td>82</td>
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<tr>
<td>Drill-Jumbo</td>
<td>105</td>
<td>Deutz F6L912W</td>
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<td>82</td>
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<td>Boom Truck</td>
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<td>79</td>
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<tr>
<td>Service-Bolting</td>
<td>413</td>
<td>Deutz F6L912W</td>
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<td>79</td>
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<td>78</td>
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<td>Perkins</td>
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<td>75</td>
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<td>Perkins</td>
<td>56</td>
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<td>Perkins</td>
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<td>Perkins</td>
<td>43</td>
<td>58</td>
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<td>Ford</td>
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<td>Ford</td>
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<td>56</td>
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<td>Personnel Carrier</td>
<td>421</td>
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<td>Utility</td>
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<td>Totals</td>
<td>34</td>
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<td>4082</td>
<td>408.2</td>
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</table>

**Table 1. Diesel Equipment Fleet**
Table 2. Current Ventilation Requirements by Activity with all Diesel Powered Equipment

<table>
<thead>
<tr>
<th>Activity/Location</th>
<th>Criterion</th>
<th>Requirement (0.063m³/s/kW)</th>
<th>Drift Velocity (3.8m x 4.0m)</th>
<th>Production/Development Shifts Total Flow Requirement (m³/s)</th>
<th>Production Shifts Total Flow Requirement (kcfm)</th>
<th>Standby Weekends/Holidays Total Flow Requirement (m³/s)</th>
<th>Average Flow Requirement (kcfm)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>m³/s</td>
<td>kcfm</td>
<td>m/s</td>
<td>fpm</td>
<td>m³/s</td>
<td>kcfm</td>
</tr>
<tr>
<td>Production Mucking</td>
<td>7yd³ Scoop</td>
<td>13.0</td>
<td>28</td>
<td>0.88</td>
<td>173</td>
<td>3</td>
<td>39.0</td>
</tr>
<tr>
<td>Production Drilling</td>
<td>3.5yd³ Scoop</td>
<td>8.0</td>
<td>17</td>
<td>0.54</td>
<td>106</td>
<td>3</td>
<td>24.0</td>
</tr>
<tr>
<td>Development</td>
<td>3.5yd³ Scoop</td>
<td>8.0</td>
<td>17</td>
<td>0.54</td>
<td>106</td>
<td>3</td>
<td>24.0</td>
</tr>
<tr>
<td>Backfill Loading</td>
<td>7yd³ Scoop</td>
<td>13.0</td>
<td>28</td>
<td>0.54</td>
<td>106</td>
<td>1</td>
<td>13.0</td>
</tr>
<tr>
<td>Rebuild Shop</td>
<td>7yd³ Scoop</td>
<td>13.0</td>
<td>28</td>
<td>0.54</td>
<td>106</td>
<td>1</td>
<td>13.0</td>
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<td>Garage</td>
<td>Various vehicles</td>
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<td></td>
<td>1</td>
<td>18.0</td>
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<tr>
<td>(1Large/1 Small, or 3 Small)</td>
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<td></td>
<td></td>
<td></td>
<td></td>
<td>1</td>
<td>18.0</td>
</tr>
<tr>
<td>Crusher &amp; Loading Pocket</td>
<td>8.0 17</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>1</td>
<td>8.0</td>
</tr>
<tr>
<td>Activity Sub Total</td>
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<td></td>
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<td></td>
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<td>139.0</td>
<td>295</td>
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<td>Other</td>
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<td></td>
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Average Flow Requirement (kcfm)
Table 3. Ventilation Requirements by Activity with all Fuel cell Powered Production Equipment, First by Remaining Diesel Requirements, and Second Including an Air Velocity Limit

<table>
<thead>
<tr>
<th>Activity/Location</th>
<th>Criterion</th>
<th>Requirement (0.063 m³/s/kW)</th>
<th>Drift Velocity (3.8m x 4.0m)</th>
<th>Production/Development Total Flow Number Oper.</th>
<th>Production Shifts Total Flow Number Oper.</th>
<th>Standby Total Flow Requirement</th>
<th>Average Flow Requirement</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>m³/s</td>
<td>kcfm</td>
<td>m/s</td>
<td>fpm</td>
<td>m³/s</td>
<td>kcfm</td>
</tr>
<tr>
<td>Production Mucking</td>
<td>Service Vehicle</td>
<td>5.0</td>
<td>11</td>
<td>0.34</td>
<td>67</td>
<td>3</td>
<td>15.0</td>
</tr>
<tr>
<td>Production Drilling</td>
<td>Service Vehicle</td>
<td>5.0</td>
<td>11</td>
<td>0.34</td>
<td>67</td>
<td>3</td>
<td>15.0</td>
</tr>
<tr>
<td>Development</td>
<td>Service Vehicle</td>
<td>5.0</td>
<td>11</td>
<td>0.34</td>
<td>67</td>
<td>3</td>
<td>15.0</td>
</tr>
<tr>
<td>Backfill Loading</td>
<td>Service Vehicle</td>
<td>5.0</td>
<td>11</td>
<td></td>
<td></td>
<td>1</td>
<td>5.0</td>
</tr>
<tr>
<td>Rebuild Shop</td>
<td>Service Vehicle</td>
<td>5.0</td>
<td>11</td>
<td></td>
<td></td>
<td>1</td>
<td>5.0</td>
</tr>
<tr>
<td>Garage</td>
<td>Various vehicles</td>
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<td>32</td>
<td></td>
<td></td>
<td>1</td>
<td>15.0</td>
</tr>
<tr>
<td>Crusher &amp; Loading Pocket</td>
<td></td>
<td>8.0</td>
<td>17</td>
<td></td>
<td></td>
<td>1</td>
<td>8.0</td>
</tr>
<tr>
<td>Activity Sub Total</td>
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<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Other</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mine Total</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Reduction</td>
<td></td>
<td></td>
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<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Activity/Location</th>
<th>Criterion</th>
<th>Requirement (0.063 m³/s/kW, or 100fpm)</th>
<th>Drift Velocity (3.8m x 4.0m)</th>
<th>Production/Development Total Flow Number Oper.</th>
<th>Production Shifts Total Flow Number Oper.</th>
<th>Standby Total Flow Requirement</th>
<th>Average Flow Requirement</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>m³/s</td>
<td>kcfm</td>
<td>m/s</td>
<td>fpm</td>
<td>m³/s</td>
<td>kcfm</td>
</tr>
<tr>
<td>Production Mucking</td>
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<td>7.5</td>
<td>16</td>
<td>0.51</td>
<td>100</td>
<td>3</td>
<td>22.5</td>
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<tr>
<td>Production Drilling</td>
<td>Minimum Velocity</td>
<td>7.5</td>
<td>16</td>
<td>0.51</td>
<td>100</td>
<td>3</td>
<td>22.5</td>
</tr>
<tr>
<td>Development</td>
<td>Minimum Velocity</td>
<td>7.5</td>
<td>16</td>
<td>0.51</td>
<td>100</td>
<td>3</td>
<td>22.5</td>
</tr>
<tr>
<td>Backfill Loading</td>
<td>Minimum Velocity</td>
<td>7.5</td>
<td>16</td>
<td></td>
<td></td>
<td>1</td>
<td>7.5</td>
</tr>
<tr>
<td>Rebuild Shop</td>
<td>Minimum Velocity</td>
<td>7.5</td>
<td>16</td>
<td></td>
<td></td>
<td>1</td>
<td>7.5</td>
</tr>
<tr>
<td>Garage</td>
<td>Various vehicles</td>
<td>15.0</td>
<td>32</td>
<td></td>
<td></td>
<td>1</td>
<td>15.0</td>
</tr>
<tr>
<td>Crusher &amp; Loading Pocket</td>
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<td>8.0</td>
<td>17</td>
<td></td>
<td></td>
<td>1</td>
<td>8.0</td>
</tr>
<tr>
<td>Activity Sub Total</td>
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<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Other</td>
<td></td>
<td></td>
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<tr>
<td>Mine Total</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Reduction</td>
<td></td>
<td></td>
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<td></td>
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</table>
### Table 4. Ventilation Requirements by Activity with all “Clean” Diesel Powered Production Equipment, First by Diesel Requirements, and Second Including an Air Velocity Limit

<table>
<thead>
<tr>
<th>Activity/Location</th>
<th>Criterion</th>
<th>Requirement (0.063m³/s/kW)</th>
<th>Drift Velocity (3.8m x 4.0m)</th>
<th>Production/Development Total Flow Requirement</th>
<th>Production Shifts Number Oper.</th>
<th>Standby Total Flow Requirement</th>
<th>Average Flow Requirement</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>m³/s</td>
<td>kcfm</td>
<td>m/s</td>
<td>fpm</td>
<td>m³/s</td>
<td>kcfm</td>
</tr>
<tr>
<td>Production Mucking</td>
<td>7 yd Scoop</td>
<td>9.1</td>
<td>19</td>
<td>0.61</td>
<td>121</td>
<td>3</td>
<td>27.3</td>
</tr>
<tr>
<td>Production Drilling</td>
<td>3.5 yd Scoop</td>
<td>5.6</td>
<td>12</td>
<td>0.38</td>
<td>74</td>
<td>3</td>
<td>16.8</td>
</tr>
<tr>
<td>Development</td>
<td>3.5 yd Scoop</td>
<td>5.6</td>
<td>12</td>
<td>0.38</td>
<td>74</td>
<td>3</td>
<td>16.8</td>
</tr>
<tr>
<td>Backfill Loading</td>
<td>7 yd Scoop</td>
<td>9.1</td>
<td>19</td>
<td>0.61</td>
<td>121</td>
<td>1</td>
<td>9.1</td>
</tr>
<tr>
<td>Rebuild Shop</td>
<td>7 yd Scoop</td>
<td>9.1</td>
<td>19</td>
<td>0.61</td>
<td>121</td>
<td>1</td>
<td>9.1</td>
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<tr>
<td>Garage</td>
<td>Various vehicles (1Large/1 Small, or 3 Small)</td>
<td>14.1</td>
<td>30</td>
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<td>14.1</td>
<td>30</td>
<td>1</td>
</tr>
<tr>
<td>Crusher &amp; Loading Pocket</td>
<td></td>
<td>8.0</td>
<td>17</td>
<td>1</td>
<td>8.0</td>
<td>17</td>
<td>1</td>
</tr>
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<td>Activity Sub Total</td>
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<td>63</td>
<td>30.1</td>
<td>64</td>
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<td>114.5</td>
<td>276</td>
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<tr>
<td>Reduction</td>
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<td></td>
<td></td>
<td></td>
<td>27%</td>
<td></td>
</tr>
</tbody>
</table>

### Activity/Location

- **Production Mucking**: 7 yd Scoop
- **Production Drilling**: 3.5 yd Scoop
- **Development**: 3.5 yd Scoop
- **Backfill Loading**: 7 yd Scoop
- **Rebuild Shop**: 7 yd Scoop
- **Garage**: Various vehicles (1Large/1 Small, or 3 Small)
- **Crusher & Loading Pocket**: 8.0

### Criterion

- **Requirement (0.063m³/s/kW)**
- **Drift Velocity (3.8m x 4.0m)**

### Production/Development Total Flow Requirement

- **Number Oper.**
- **Total Flow Requirement**

### Production Shifts Number Oper. Total Flow Requirement

- **Total Flow Requirement**

### Standby Total Flow Requirement

- **Total Flow Requirement**

### Average Flow Requirement

- **Total Flow Requirement**

### Activity Sub Total

- **Total Flow Requirement**

### Reduction

- **Total Flow Requirement**

### Other

- **Total Flow Requirement**

### Mine Total

- **Total Flow Requirement**

### Reduction

- **Total Flow Requirement**
Table 5. Ventilation Attributable CO₂ Greenhouse Gas Emissions Related to Energy Usage Upon the Adoption Of Fuelcells Or “Clean” Diesels

<table>
<thead>
<tr>
<th>Scenario</th>
<th>Average Airflow</th>
<th>Electricity</th>
<th>Propane</th>
<th>Diesel</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>m³/s Reduction</td>
<td>Usage MWh %</td>
<td>Usage ML %</td>
<td>Usage ML %</td>
</tr>
<tr>
<td><strong>Current Practice</strong></td>
<td>143 20%</td>
<td>5869 30%</td>
<td>1179 20%</td>
<td>1.80 0.72</td>
</tr>
<tr>
<td><strong>Fuel Cell Option</strong></td>
<td>112 22%</td>
<td>4090 40%</td>
<td>818 30%</td>
<td>2.2 0.18</td>
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<tr>
<td><strong>“Clean” Diesel Option</strong></td>
<td>120 16%</td>
<td>4388 40%</td>
<td>878 30%</td>
<td>1.7 0.72</td>
</tr>
<tr>
<td><strong>Total CO₂</strong></td>
<td>1183 22%</td>
<td>820 40%</td>
<td>820 30%</td>
<td>2.0 31%</td>
</tr>
</tbody>
</table>

Table 6. Comparative Breakdown of Ventilation Costs and Savings Upon the Adoption of Fuelcells or “Clean” Diesels

<table>
<thead>
<tr>
<th>Scenario</th>
<th>Airflow Reduction</th>
<th>Primary Surface Fans</th>
<th>&quot;Production&quot; Auxiliary Fans</th>
<th>Heating</th>
<th>Total</th>
</tr>
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<tbody>
<tr>
<td></td>
<td>m³/s %</td>
<td>Cost $000s %</td>
<td>Cost $000s % % Total</td>
<td>Cost $000s % % Total</td>
<td>Cost $000s % % Total</td>
</tr>
<tr>
<td><strong>Current Practice</strong></td>
<td>143 20%</td>
<td>207 21%</td>
<td>188 19%</td>
<td>576 59%</td>
<td>971 22%</td>
</tr>
<tr>
<td><strong>Fuel Cell Option</strong></td>
<td>112 22%</td>
<td>108 48%</td>
<td>166 15%</td>
<td>451 23%</td>
<td>925 20%</td>
</tr>
<tr>
<td><strong>“Clean” Diesel Option</strong></td>
<td>120 16%</td>
<td>128 38%</td>
<td>166 15%</td>
<td>483 16%</td>
<td>777 20%</td>
</tr>
</tbody>
</table>
B. CREIGHTON MINE – INCO LIMITED, ONTARIO DIVISION

1. VENTILATION REQUIREMENTS AND DESIGN CRITERIA

This underground operation produces 4,000 tons/day from a near continuous operation comprising a schedule of two back-to-back 10 hr shifts per day, with a 4 hr blast clearance period during the night. On each weekend day, one shift is non-productive. Consequently this mine operates at peak ventilation demand 85% of the time and in the remainder it employs some ventilation cost management through turning down, or off, a portion of its primary ventilation fans. Currently, approximately 2,500 tons/day are extracted from the upper mine (above 2100 m depth) through a vertical retreat mining (VRM) method, and 1,500 tons/day are extracted from the upper levels of a new deep orebody (between 2100 and 2500 m depth) through a combination of VRM and cut-and fill (C&F) methods. In the future, this mine’s primary production is currently planned to be solely from the deep orebody and could be exclusively by a C&F method. The typical drift dimensions in this mine are 4.9 m wide by 4.5 m high.

Figure 1 - Ventilation Schematic of a Large Sized Base Metal Operation.

Figure 1 shows the general layout of the mine’s ventilation system from the surface through to the future bottom of the mine, 2500 m below surface. The mine has three main intake airways; Shaft A from the surface down to 2100 m level, the main fresh air raise (MFAR) and a ramp.
system, which ultimately delivers air to 2200 m level and the internal Shaft B taking air from 390 m level to 2200 m level. The deep orebody has a single main intake fresh air raise (MFAR) and an extension that will eventually deliver air to 2500 m level. To exhaust the air the mine predominantly uses a main return air raise (MRAR) from the lower orebody and a dedicated ventilation shaft, Shaft C, back to surface. To provide the flow through this system and to the required depth, the mine employs 11(or 13) primary and booster fans, the majority of which, eight (or ten) in parallel arrangements of two fans, are in the intake system. The remaining three fans, also in parallel, are at the top of the exhaust shaft.

Fresh air to the production areas is drawn off of the intake system usually under the assistance of auxiliary fans. Auxiliary ducting then delivers this airflow to the desired production areas. From the production areas the return airflow is exhausted through the production levels to the MRAR or return shaft.

In this mine, the current ventilation design criteria is a combination of diesel and heat requirements, however in the future, the primary flow to the production areas will be exclusively dictated by heat rejection needs, but local auxiliary flows could still be based upon diesel exhaust dilution needs.

In the subsequent analysis of this mine, the potential benefits of fuelcells will be explored initially under the current conditions but also for the future requirements when flows are dictated by heat concerns.

1.1 Ventilation Design Criteria for the Upper Area and the Deep Orebody

In the current ventilation system, with the majority of production above 2100 m, the controlling design criterion is diesel exhaust based upon the rated output power of diesel equipment. This mine uses 0.079 m³/s per rated kW of diesel power in its overall primary design. This is higher than the Provincial requirement of 0.063 m³/s/kW at the point of application to take account of leakage and also in recognition of increasing heat rejection issues. Secondary considerations are blast fume clearance and dust control.

As a consequence of the mine’s adopted diesel design criteria the mine uses the following flow allocation:
- Minimum airflow for active stopes: \( Q_{stope} = 336 \text{ kW} \times 0.079 \text{ m}^3/\text{s/kW} = 26.5 \text{ m}^3/\text{s} \)
- Minimum airflow for development headings: \( Q_{dev} = 243 \text{ kW} \times 0.079 \text{ m}^3/\text{s/kW} = 19.2 \text{ m}^3/\text{s} \)
- Minimum airflow for haulage ramps: \( Q_{ramp} = 354 \text{ kW} \times 0.079 \text{ m}^3/\text{s/kW} = 28 \text{ m}^3/\text{s} \)

In the ventilation simulations of this mine minimum flows allowed were 15 m³/s on inactive levels to permit service vehicles access and 11.5 m³/s to maintain a minimum velocity of 0.5 m/s in non-vehicle access areas. As a consequence of these flow allocations, the current mine ventilation system was simulated with a primary flow of 608 m³/s.
1.2 Ventilation Design Criteria for the Deep Orebody

For the deep orebody, the mine has determined through climatic modelling the airflow requirements for each production level. Figure 2, shows the increasing intake airflow temperature and minimum airflow requirement for each particular level of the deep orebody, as a function of production rate, upon taking into account the heat generated from strata, and the mining machinery involved in development and production processes.

Consequently, the flow requirements in the lower mine (the deep ore body), will increase from 0.099 m³/s/tpd at 2100 m to 0.170 m³/s/tpd at 2500 m, an increase of 72%. In this analysis, the potential benefits of fuelcells was determined when all the future production will be between 2450 m to 2500 m, and similar to the current analysis, minimum flows will be maintained on all inactive levels. Under these conditions the mine ventilation system was analyzed with a primary flow of 652 m³/s for diesel based production.

![Airflow Requirements and Intake Airflow Temperatures as a Function Of The Mining Depth](image)

Figure 2 - Airflow Requirements and Intake Airflow Temperatures as a Function of the Mining Depth.
2. VENTILATION SYSTEM OPERATING COST USING DIESEL POWERED EQUIPMENT

2.1 Current Situation – based upon diesel exhaust criteria

Ventilation modelling and simulation exercises of the current situation have shown that in order to deliver the required air volumes to the production areas and throughout the mine, an intake flow of 608 m\(^3\)/s is needed during production. This requires the mine’s main and booster fans to develop a total pressure of 16.708 kPa, at a cost of $3.1M(Can)/year. Across the weekends the flow drops to 558 m\(^3\)/s at a combined total pressure of 14.011 kPa and a cost of 0.3M(Can)/year. Base upon electricity charges of $0.055/kWh, the combined annual operating power cost of the primary ventilation system is $3.4M(Can)/year.

For the secondary distribution of ventilation, the mine uses approximately 200 auxiliary fans with motor powers ranging from 37 kW (50 hp) through to 112 kW (2 x 75 hp). The combined installed power of these fans is 6,197 kW and it has been assumed that these fans when operating are working at their rated power. For two shifts during the weekend, the mine estimates that 50% of the auxiliary ventilation system fans are turned off, and during statutory holidays all auxiliary systems should be off. Based upon this information the auxiliary ventilation operating power cost in the current system have been estimated as $1.5M(Can)/year.

At this mine, heating of the mine air during winter is predominantly through natural effects and consequently the mine’s consumption and hence cost of fossil fuels for heating is minimal in comparison to the electrical costs. Therefore the mine’s annual ventilation operating costs effectively comprise the electrical costs for the primary and auxiliary ventilation system’s, namely $4.9M/yr for an electricity consumption of 89.1 GWh.

2.2 Future Requirements – based upon heat management considerations

Using diesel powered mobile equipment, modelling and simulation exercises have shown that the mine will have to supply 652 m\(^3\)/s, an increase of 7%, when all the production is from the lower three levels of the mine and to ensure minimum flows throughout the rest of the operation are maintained. For this flow and its distribution, ventilation simulations indicate that the mine’s main and booster fans will have to develop a combined total pressure of 19.95 kPa with a resultant annual operating power cost of $4.2M(Can)/year, an increase of 23%.

For the auxiliary ventilation distribution at this future point it has been estimated that the mine’s auxiliary fan power requirement would increase to 7,436 kW with a resultant annual operating cost of $1.8M(Can)/yr, an increase of 20%. This increase is due to larger auxiliary fans needed to assist in overcoming additional pressure losses along the ventilation raises.

Therefore, upon neglecting the relatively minor cost of heating, the mine’s future ventilation operating cost would be $6.0M(Can)/year, for an electricity consumption of 109.1 GWh.
3. VENTILATION SYSTEM OPERATING COST USING FUELCELL POWERED EQUIPMENT

3.1 Current Application – based upon fuelcell airflow requirements

Determining the appropriate ventilation requirements in a mine employing fuelcells presents some difficulty, depending upon the degree of conversion and on how the light-duty diesel vehicles plus other secondary requirements are accounted for in the original design.

However, assuming that the 4,000 tpd are produced from eight active stopes within the upper area of the mine and using the 0.042 m³/s/tpd plus 25% additional allowance for miscellaneous needs, and maintaining minimum flows elsewhere, the mine would require total intake airflow of 459 m³/s, which represents a 24% reduction when compared to the diesel based requirement. To deliver and distribute this airflow during the production period, ventilation simulations show that the required total power of the primary and booster fan system drops to 6,232 kW, with an associated cost of $1.63M/yr, a 51% reduction. For the weekends and statutory holidays, the magnitude of reduced air volumes and subsequent costs would not change. Base on electricity charges of $0.055/kWh, the overall annual cost of the primary ventilation system would be $1.77M(Can)/year, a 48% reduction.

Operating costs in the secondary ventilation system are even more difficult to estimate. If it where possible for the mine to downsize its current inventory of secondary fans then savings would result. A reduction would probably be possible for the short draw point mucking locations of the VRM method, however, it may not be possible for the C&F mining method where the potential for a high volume blast clearance would have to be maintained. However, if the mine’s total auxiliary fan power could be reduced by approximately 15% the operating cost would drop to $1.27M(Can)/year. Therefore if fuelcells were introduced immediately to this operation the combination primary and auxiliary ventilation system operating power costs would be $3.04M(Can)/year for an electricity consumption of 55.3 GWh, for a combined 38% reduction.

3.2 Future Situation – based upon fuelcell airflow requirements

In the future the airflow requirements for the deep orebody will be determined upon heat rejection demands of the hotter strata at depth. The mine has already determined these requirements for a diesel operation based upon production rates. However, to determine a comparable demand for fuelcells several assumptions have to be made. The diesel-based demands have been determined by means of climatic modelling. This modelling showed that increasing volumes are required at depth to combat the increasing component of heat from the strata although the input from machinery would tend to stay the same. To obtain the needs of fuelcells, the relative machinery and non-machinery heat components at varying depths was determined from the mine’s production rate flow requirements and knowledge of the thermal parameters within the mine using an underground climatic simulator. Once the relative machinery and non-machinery components were determined, a simplified analysis on replacing diesel machinery with fuelcell units was performed as follows:
An underground climatic model has been developed using Climsim™ simulator for a 206 Diesel LHD operating in a typical dead-end development (4.9 m wide by 4.5 m high) at 2200 m depth. Climatic simulations show that of the total heat load of 392 kW added to the air, 206 kW (52%) is generated by the diesel LHD and 186 kW (48%) is generated by non-equipment heat sources. Upon replacing the diesel powered LHD by a 206 kW fuelcell powered unit, the total heat load decreases to 275 kW, of which 89 kW (33%) is generated by the fuelcell LHD and 186 kW (77%) is still generated by non-equipment heat sources. For both diesels and fuelcells the intake airflow requirements were determined such that ambient wet bulb temperature were maintained below 28°C. Then, by varying the VRT and the barometric pressure the required intake airflows during the mining cycle have been determined for both diesel and fuelcell powered LHDs as a function of the mining depth. Figure 3 shows the relative reduction that might be possible from employing fuelcells at increasing depth as derived from the respective diesel/fuelcell airflow requirements.

Although this analysis may not be precise, it does indicate that with increasing depth the heat from non-machinery sources increases. A point is reached where for heat removal purposes whether a mine is using a fuelcell or diesel based equipment, it makes little difference to the airflow requirements.
However, to provide an indication of the savings for this mine when all the production is from the lower mine (between 2450-2500 m), Figure 3 has been used to adjust the production based flow requirements. Consequently, ventilation simulations maintaining minimum flows elsewhere show that the mine would require intake airflow of 592 m$^3$/s with fuel cells, a 9% reduction compared to airflow requirements for diesels. To deliver this airflow to the production areas, the mine’s primary and booster fan system’s operating cost drop to $3.1M(Can)/year, a 26% reduction compared to the diesel scenario. Again for the weekends and statutory holidays, the magnitude of reduced air volumes and subsequent costs would not change. Therefore the overall annual operating power costs of these fans would be $3.3M(Can)/year, a 23% reduction.

By downsizing the secondary production fans, the mine’s total auxiliary fan power may be reduced by approximately 12%, with a similar reduction in operating cost to $1.6M(Can)/year. In combination the primary and auxiliary ventilation system operating cost would be $4.8M(Can)/year, for an electricity consumption of 87.9 GWh, which represent a 19% reduction.

4. ENVIRONMENTAL BENEFITS

For Creighton mine’s current situation (majority of production coming from the upper levels), and future situations (production coming from the deep orebody), a comparison of greenhouse gas emissions (GHG) expressed in tonnes of equivalent CO$_2$ emissions per year for diesels versus fuelcells is given in Table 1 and table 2, respectively. Upon replacing diesels with fuelcells, the combined ventilation electricity consumption, which includes the primary, auxiliary and cooling systems, reduces by 38% for the upper orebody scenario and by 19% for the deep orebody scenario. At this operation, within the fuelcell option approximately 25% of the remaining non-production small vehicles would still be powered by diesel engines. As result, during the mining cycle diesel fuel consumption reduces by approximately 75%. The conversion factors used were 200 tonnes of equivalent CO$_2$ emissions per 1GWh of generated electricity (this is specific to Canada and reflects the make-up of the country’s electrical generation primary from hydro-electric stations), 1.90 tonnes/ML of natural gas, and 2.777 tonnes/ML of diesel fuel.
Table 1 - Equivalent CO₂ GHG Emissions For Fuelcells vs. Diesels – Creighton (Upper Orebody)

<table>
<thead>
<tr>
<th>Scenario</th>
<th>Airflow</th>
<th>Electricity</th>
<th>Natural Gas</th>
<th>Diesel Fuel</th>
<th>Total GHG</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Intake Airflow</td>
<td>Usage</td>
<td>CO₂ Equivalent Emissions</td>
<td>Usage</td>
<td>CO₂ Equivalent Emissions</td>
</tr>
<tr>
<td>Current Practice</td>
<td>608</td>
<td>89,091</td>
<td>17,818</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Fuelcell Option</td>
<td>459</td>
<td>55,273</td>
<td>11,054</td>
<td>-</td>
<td>-</td>
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<tr>
<td>GHG Reductions</td>
<td></td>
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<td></td>
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</tbody>
</table>

Based upon these conversion factors, it can be seen that at this operation the fuelcell technology would reduce the mine’s attributable GHG emissions through reduced electricity, and diesel fuel consumption by 6,766 tonnes of equivalent CO₂/year for the upper orebody scenario (current application) and 9% for the deep orebody scenario (future requirements), which represents a 38% and a 9% reduction, respectively.

Table 2 - Equivalent CO₂ GHG Emissions For Fuelcells vs. Diesels – Creighton, Deep Orebody

<table>
<thead>
<tr>
<th>Scenario</th>
<th>Airflow</th>
<th>Electricity</th>
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5. SUMMARY

This operation employs two ventilation design criteria, diesel exhaust within the upper areas of the mine (surface – 2100 m depth) and heat management within the deep orebody (between 2100 – 2500 m depth). Within the upper areas this mine uses 0.079 m³/s per rated kW of diesel power in its overall primary ventilation design, which is higher than the provincial requirement of 0.063 m³/s/kW in order to take account of leakage and increased heat generation issues. For the deep orebody, the airflow requirements have been determined through climatic modelling. The airflow requirements within the deep orebody will increase from 0.099 m³/s/tpd at 2100 m depth to 0.170 m³/s/tpd at 2500 m depth.

Under the current scenario, these analyses show potential for an overall 38% reduction in the combined electrical costs of the primary and auxiliary ventilation system for associated average intake airflow reductions of 24%. Reductions in the electrical operating costs of the primary ventilation system are in the order of 51%, whether reductions in the electrical operating cost of the production operating fans of the auxiliary ventilation system are in the order of 15% due to their inflexibility to accommodate reduced airflow regimes during the mining cycle and the need to maintain a high volume flush for blast fume clearance. The GHG emissions produced by this underground operation under the current scenario are 17,821 tonnes/year of equivalent CO₂. By replacing diesels with fuelcells, the potential GHG emissions reductions are in the order of 6,766 tonnes/year of equivalent CO₂, which represents a 38% reduction.

Under the future ventilation requirements, these analyses show potential for 23% reduction in the combined electrical costs of the primary and auxiliary ventilation systems. Reductions in the electrical operating costs of the primary ventilation system are in the order of 26%, whether reductions in the electrical operating costs of the auxiliary ventilation system are in the order of 12%. The GHG emissions produced under the future ventilation scenario are 21,820 tonnes/year of equivalent CO₂. By replacing diesels with fuelcells the GHG emissions are reduced to 17,580 tonnes/year of equivalent CO₂, which represents a 9% reduction.

This operation employs a natural heating/cooling system, therefore heating/cooling cost are presently negligible compared to the electrical operating costs of the primary and auxiliary ventilation system.

The comparative intake airflow and combined ventilation operating cost for fuelcells compared to diesel-powered equipment is given in Figures 4 and 5.
Figure 4 - Total Mine Intake Airflow – Diesels vs. Fuelcells

Figure 5 - Combined Ventilation Operating Cost – Diesels vs. Fuelcells
C. MCCREEDY EAST MINE– INCO LIMITED, ONTARIO DIVISION

1. INTRODUCTION

The McCreedy East mine, owned and operated by Inco Limited is located in Sudbury, approximately 400 km north of Toronto. McCreedy East is one of several mines in the Sudbury basin producing not only nickel but also significant amounts of copper, precious metals and other by-products. McCreedy East mine has three distinct orebodies:

- Lower Coleman Orebody – employing bulk mining methods (VRM), mining method extensively used at Inco’s other operations
- Main Orebody – employing large scale cut-and-fill post-and-pillar mining methods, and
- 153 Orebody – which requires a very small-scale cut-and-fill mining method.

The mine produces 4,000 tons/day, of which approximately 1,300 tons/day are extracted from the 153 Orebody using narrow vein mining methods (i.e. small scale mechanized cut-and-fill) and 2,700 tons/day extracted from the Main and Lower Coleman Orebody using large scale cut-and-fill mining method. Although small by Inco’s standards, production from the 153 Orebody accounts for more than half of the company’s entire platinum-group metal output from Sudbury. It also produces significant amounts of copper, nickel, silver and gold.

Vein thickness within the 153 Orebody ranges from less than 125 mm up to 24 meters, hosted within a package of felsic gneisses and breccias in the footwall of the Sudbury Basin. The orebody has a strike length of 365 meters and is currently being mined between 4250 and 4800 levels. The veins are composed of mainly chalcopyrite with accessory minerals such as yellow-gold coloured millerite, which grades 64% nickel and blue-green borite, which grades 63% copper.

The mine has 386 employees, with an operational schedule of 12 hours/shift, 2 shifts/day. The Lower Coleman orebody is accessed by the 6.2 m diameter concrete lined production shaft from the surface to the 3370 level and a ramp system. The Main orebody is being accessed by the 5.8 m diameter concrete lined McCreedy East No.1 Shaft from surface to 3770 level and by a ramp system between the 3510 and 4315 levels. The 153 Orebody is accessed by the same McCreedy East No. 1 Shaft from surface to the 3770 main level and a ramp system between levels 3770 and 4810 (See Figure 1).

Within the Main and Lower Coleman orebodies, typical stope dimensions are 8.5 m wide by 9.8 m high. Typical drift development is 4.8 m wide by 5.5 m high. Drilling is performed with 64 kW (86 hp) two-boom diesel electric jumbo-drills. Mucking is performed with 8 yd³ bucket 231 kW (310 hp) and 5 yd³ bucket 200 kW (270 hp) Elphinstone LHDs.

Drifts on the 153 Orebody are normally 2.6 m high, but up to 3.4 m high at access points to accommodate the auxiliary ventilation ducts. Stopes can be up to 7.3 m wide, depending on the width of the vein. Drilling is performed with Atlas Copco’s “Boomer 104” 22 kW (29 hp) single boom drill rigs, specifically designed for narrow vein mines. Mucking is performed with 2.5 yd³ 87 kW (116 hp) JCI diesel LHDs. From the ore passes, the ore is then handled by the 50-ton
electrical Kiruna Trucks along the main haulage to the dumping stations located on the 3770 level, and from here to the surface by Coleman Mine production shaft’s hoisting system.

The basic layout of the mine is shown in the ventilation Schematic, Figure 1. Presently, the mine’s total intake airflow is 538 m$^3$/s, of which 396 m$^3$/s is being delivered through the 5.8 m diameter concrete lined McCreedy East No. 1 Shaft and 142 m$^3$/s through the 6.2 m diameter concrete lined Coleman Production Shaft.

Fresh air to the 153 Orebody is delivered through the 5.8 m diameter McCreedy East No. 1 Shaft, the 153 Fresh Air Ramp and the internal raise system connecting 153 Orebody’s production levels under the assistance of two Alphair 112-66-880 (1500 hp) surface fans. Fresh air to the production areas is then drawn off of this internal raise system that takes the air down to 153 Orebody’s 4810 level. Auxiliary ducting then takes this air to the desired production locations under the assistance of the auxiliary fans. In the production area series ventilation is employed. The return air is then exhausted from the bottom of the 153 Orebody through an exhaust raise system (between levels 4810 and 4250), to Levack Return Air Drift and through the 153 Orebody’s main haulage ramp and two 1.2 m diameter ventilation ducts installed inside the 5.8 m diameter McCreedy East No. 1 Shaft, under the assistance of two 112 kW (150 hp) Joy 48-26-1770 return air fans in a parallel arrangement (See Figure 1).

Fresh air to the Main Orebody is delivered through the McCreedy East No.1 Shaft and the Production Shaft under the assistance of two 75 kW (100 hp) Alphair 72-26-870 surface fans in a parallel arrangement, the main orebody’s internal fresh air raise system, connecting the 4315, 4050, 3770, 3575 and 3510 production levels. Fresh air to the production areas is drawn off of this internal raise system naturally or under the assistance of auxiliary fans. The return airflow is then exhausted through the Main orebody’s exhaust raise system, the Kiruna Truck Haulage Ramp and Coleman Mine’s Main return air raise system (Main RAR) under the assistance of two 150 kW (200 hp) Joy 66-30-1170 surface fans installed in parallel at the top of Mai RAR and a 95 kW (125 hp) Joy 66-26-1170 fan installed at the top of the 3000 return air raise (3000 RAR).

For ground control, the mine employs a combination of sandfill and plain rock fill. The sandfill is delivered to the mined out cut-and-fill stopes through a piping network from the surface at a rate of 2000 tons/day, the plain rock fill is placed into the empty stopes by the mines production scoops.
2. CURRENT VENTILATION REQUIREMENTS AND DESIGN CRITERIA

In the current ventilation system, with the majority of production coming from the Main orebody and the 153 Orebody, the controlling ventilation design criterion is diesel exhaust based upon the rated output power of diesel equipment.

This mine uses 0.079 m$^3$/s per rated kW of diesel power in its overall primary design. This is higher than the requirement in the Province of Ontario of 0.063 m$^3$/s per rated kW of diesel power, in order to take account for leakage especially within the 153 Orebody’s ventilation system. Secondary considerations are blast fume clearance and dust control. There are no other strata gas issues such as methane that must be considered into the design criteria.

In the ventilation simulations of this mine, minimum air volumes of 15 m$^3$/s were required in order to allow access of service vehicles and approximately 12 m$^3$/s to maintain minimum airflow velocities of 0.5 m/s along the major development and haulage drifts.

3. VENTILATION SYSTEM OPERATING COST USING DIESEL POWERED EQUIPMENT

Ventilation modelling and simulation exercises of the current ventilation system have shown that in order to deliver the required air volumes to the production areas and throughout the mine, a total intake air volume of 538 m$^3$/s is needed. From this total intake air volume, 396 m$^3$/s is being delivered down the 5.8 m diameter McCreedy East No.1 Shaft and 142 m$^3$/s down the Coleman Orebody’s Production Shaft. The return airflow from the production areas of the 153 Orebody and the Main Orebody, is then exhausted through two 1.2 m diameter ventilation ducts, installed inside the 5.8 m diameter McCreedy East No.1 Shaft and Levack Mine’s return air raise system. In order to maintain minimum airflow velocities, approximately 80 m$^3$/s of fresh air is being also transferred from the Main Orebody to the Kiruna Truck Haulage Ramp, which is then exhausted to the surface through Coleman Orebody’s main return air raise system (Main RAR). This requires the mine’s main and booster fans to develop a total pressure of 6730 Pa, with a combined fan power of 4828 kW. Based upon electricity charges of $0.065/kWh, the primary ventilation system’s operating cost is $2,750,000/year.

The mine’s secondary ventilation system within the 153 Orebody, the Main orebody and Lower Coleman orebody uses various fans with motor powers ranging from 15 kW (20 hp) to 112 kW (150 hp). To deliver the required air volumes to the production areas, the mine’s auxiliary fans need to develop a combined total power of 3,896 kW, with an associated operating power cost of $2,200,000/year.

In addition to the primary and auxiliary ventilation systems operating power costs, the mine also spent another $350,000/year on natural gas to heat its intake air during the winter season.

Therefore, the combined electrical power cost for the primary ventilation, auxiliary ventilation and heating systems is: $2,750,000/year (primary ventilation) + $2,200,000/year (auxiliary ventilation) + $350,000/year (heating) = $5.3M/year.
4. VENTILATION SYSTEM OPERATING COST FOR FUELCELL POWERED EQUIPMENT

During the mining cycle, assuming that fuelcell powered equipment will replace the primary diesel equipment, and based upon the mines production rate of 4,000 tons/day and a 90 cfm/tpd plus 25% additional airflow allowance for miscellaneous needs, the mine would require an intake airflow of 212 m$^3$/s. In order to maintain minimum airflow velocities along the major haulage routes and within the inactive areas of the mine, as well as to provide a sufficient air volume for the garage and the chute loading areas, an additional 220 m$^3$/s would be required. Therefore the mine’s total intake airflow is 432 m$^3$/s, which represents a 20% reduction when compared to the diesel based requirements.

To deliver and distribute this air volume to the active areas of the mine, ventilation simulations show that the required total pressure of the primary ventilation system drops to 4,330 Pa, with a combined surface and booster fan power of 2,494 kW and associated power costs of $1,384,500/year, a 50% reduction.

Operating costs in the auxiliary ventilation system are more difficult to be evaluated, because unless the mine changes the size of the auxiliary fans, the auxiliary system would be unable to accommodate reduced airflow regimes. Further to this, due to the cut-and-fill mining method employed at both 153 Orebody and the Main orebody, a higher volume of flush may be required to remove blast fumes, therefore, downsizing all secondary production fans may not be an option. However, to accommodate reduced airflow needs, if the mine would downsize some of its 112 kW (150 hp) production fans to the next lowest size in the inventory (i.e. 75 kW), the auxiliary system’s total fan power would be reduced to 3,188 kW, with associated power costs of $1,769,700/year, a 19% reduction.

As a consequence of a 20% reduction in the mine’s total intake airflow, the heating costs during the winter season would similarly be reduced by 20% to $280,000/year. In combination, to supply 432 m$^3$/s fresh air to the working areas and main haulage routes based upon airflow requirements for fuelcell powered equipment, the combined energy operating cost would be: $1,384,500/year (primary system) + $1,769,000/year (secondary system) + $280,000/year (heating) = 3.433M/year, an overall 36% reduction.

5. ENVIRONMENTAL BENEFITS

In Table 1, a comparison is given in green house gas emissions (in equivalent tonnes/year CO$_2$ emissions), at McCreedy East mine for diesels versus fuelcells, based upon their ventilation requirements as previously described. Upon replacing diesels with fuelcells, the combined ventilation electricity consumption at this operation reduces by 36%, the natural gas usage by 20% and the diesel consumption by approximately 75%. Within the fuelcell option, approximately 25% of the remaining non-production small vehicles would still be powered by diesel engines. The conversion factors used were 200 tonnes of equivalent CO$_2$ per GWh of generated electricity, (this is specific to Canada and reflects the make-up of the country’s
electrical generation primary from hydro-electric stations), 1.90 tonnes/ML of natural gas, and 2.777 tonnes/ML of diesel fuel.

Based upon the conversion factors, it can be seen that at this operation the fuelcell technology would reduce the mine’s attributable GHG emissions by 9,707 tonnes of equivalent CO$_2$ per year, which represents a 36% reduction if compared to the diesel option.

**Table 1 - CO$_2$ Equivalent GHG Emissions Reductions Upon the Adoption of Fuelcells**

<table>
<thead>
<tr>
<th>Scenario</th>
<th>Airflow</th>
<th>Electricity</th>
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<th>Diesel Fuel</th>
<th>Total GHG</th>
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<tr>
<td></td>
<td>Intake Airflow m$^3$/s</td>
<td>Usage MWh</td>
<td>CO$_2$ Equivalent Emissions T/year</td>
<td>Usage ML</td>
<td>CO$_2$ Equivalent Emissions T/year</td>
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<td>Fuelcell Option</td>
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<td>GHG Reductions (tonnes/year)</td>
<td></td>
<td></td>
<td></td>
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</table>

6. **SUMMARY**

The analyses show potential for an overall 36% reduction in the combined electrical and heating costs of the primary and auxiliary ventilation system for an associated intake airflow reduction of 20%. Reductions in the electrical operating costs of the primary ventilation system are quite significant, in the order of 50%, the reductions in the electrical operating cost of the production fans within the auxiliary ventilation system are in the order of only 19% due to their inflexibility to accommodate reduced airflow requirements during the mining cycle.

Based upon this analysis, it can be seen that fuelcells can have in important economic impact upon the mine’s ventilation system operating power consumption. They also have a significant impact in reducing GHG emissions. This can depend upon any energy savings from hydrogen generation. The comparative intake airflow and combined ventilation operating cost for fuelcells compared to diesel-powered equipment is given in Figure 2 and 3.
Figure 2 - Mine Total Intake Airflow – Diesels vs. Fuelcells

Figure 3 - Combined ventilation Operating Costs – Diesels vs. Fuelcells
D. LARONDE MINE – AGNICO EAGLE LTD.

1. INTRODUCTION

Mining at Laronde started in 1988, and two years later a long-term exploration program was initiated on both surface and underground, resulting in the discovery of Shaft No. 2 and Penna Shaft ore zones. The Penna Shaft provides access to 51.3 million tones of reserves containing approximately 6 million ounces of gold. The current gold reserve makes Laronde one of the largest underground gold deposits in Canada.

Traditionally, the Laronde operation only mined gold with silver and copper by products, but in 1998, a zinc circuit has also been commissioned. The mill on-site now produces gold and silver bars and copper and zinc concentrates.

The Penna Shaft is actually the deepest shaft of all underground operations in the Province of Quebec. In fact, at 2,250 m (7380 feet) depth, the Penna Shaft is one of the deepest single-lift shafts in North America. In order to accommodate the extreme depth of the shaft and the production increase (from 7,000 tons/day, up from the original 2,000 tons/day), both the hoisting facility and the headframe have undergone major modifications. The headframe height was increased by approximately 10.5 m to a total height of 60.5 m. The hoist drum diameter was also increased from 4.9 m to 5.8 m and its motor power was doubled from 4,000 hp to 8,000 hp.

The mine has 488 employees (222 surface/266 underground) with an operation schedule of 8 to 12 hours/shift, 2 shifts/day (8 hours/dayshift for the engineering/admin staff, 10 hours/shift for the underground employees and 12 hours/shift for the mill workers).

The new Penna Shaft orebody is 854 - 2,250 m below the surface and is accessed by two 6.7 m diameter concrete lined shafts and a ramp system in the upper part of the mine (between levels 110 – 150), and the lower part of the mine (between levels 170 – 290). The mining method employed at the Laronde Mine is long-hole stoping. Typical stope dimensions are 25 m high, 15m wide and 3 to 30 m long. Typical development dimensions are 4.2 m wide and 4.5 m high.

Drilling is performed with 61 kW (82 hp) Atlas Copco H-227 and 39 kW (52 hp) Tamrock Data-Solo diesel-electric drill jumbos powered by Deutz F6L912W engines. Mucking is performed with 8 yd³ bucket 224 kW (300 hp) Tamrock Toro and 7 yd³ bucket 175 kW (235 hp) Tamrock AJC-210 diesel LHDs powered by Detroit Diesel Series 50 diesel engines and 26 ton Tamrock Trucks powered by 224 kW (300 hp) Detroit Diesel Series 60 diesel engines. For ground control the mine employs a combination of paste fill, cemented rock and plain rockfill. The paste fill is delivered to the empty stopes through a piping network from surface, whether the cemented and plain rockfill is placed into the empty stopes by the mines “production scoops” at a rate of 4800 tpd for the paste fill and 1000 tpd for the cemented and plain rockfill, respectively.

The basic layout of the mine is shown in the ventilation schematic, Figure 1. The ventilation system for the Penna Shaft orebody has two separate systems. The first system is from the
surface to level 149, and the second from level 149 to level 220, the bottom of the mine. The mine has two main intake airways; the main fresh air raise (Main FAR) and Penna Shaft.

Presently the mine’s total intake fresh air is 703 m$^3$/s, of which 103 m$^3$/s is being delivered to the working areas through the Penna Shaft and 600 m$^3$/s through the 6.7 m diameter Main FAR under the assistance of 2 TLT, 2.65 m diameter fans with variable speed drives and 750 kW (1000 hp) motors. Air to the production areas is drawn of this intake system either naturally or under the assistance of booster fans. Auxiliary ducting then takes the air to the desired production locations under the assistance of the auxiliary fans. The return airflow is then exhausted from the bottom of the mine through a 3.4 m diameter and a 4.6 m diameter exhaust raise to Level 149, then through the 6.7 m diameter main return air raise (Main RAR) to the surface under the assistance of 2 TLT, 2.65 m diameter surface fans with variable speed drives and 1300 kW motors.

![Ventilation Schematic of Laronde Mine, province of Quebec](image)

In order to provide good working conditions within the lower levels of Penna Shaft’s orebody, a 2-unit 1.65 MW/unit cooling system has been installed on level 146. Another 1.65 MW cooling unit will be installed on the same level in July-August, 2003, followed by the installation of a 13 MW surface cooling system. This 3-unit 5 MW underground refrigeration system from level 146 will provide cooling especially for the production areas on levels 194, 200 and 215.
2. CURRENT VENTILATION REQUIREMENTS AND DESIGN CRITERIA

In the current system, the controlling criterion for ventilation design is diesel exhaust dilution. Secondary considerations for the lower levels of Penna Shaft working areas are heat (generated from strata auto-compression and mining equipment), blast fume clearance and dust control. Due to the ore/strata configuration of the mine, there are no other strata gas issues such as methane that must be considered into the design criteria.

For underground operations in the Province of Quebec where intake airflow requirements for development and production operations are determined according the diesel exhaust criteria, the “clean” diesel engine technology has also been taken onto consideration based upon the Canadian Safety Association Standards. Laronde mine uses 0.0475 m\(^3\)/s per rated kW of diesel power in its overall primary and auxiliary ventilation design for the upper levels of the mine (between levels 65 –149) and 0.053 m\(^3\)/s per rated kW of diesel power for the lower levels (between levels 149 – 220), in order to take account for increased leakage and heat rejection issues. This is 25% and 15%, respectively, lower than the air volume requirements in the Province of Ontario of 0.063 m\(^3\)/s per rated kW of diesel power.

3. VENTILATION SYSTEM OPERATING COST USING DIESEL POWERED EQUIPMENT

Ventilation modelling and simulation exercises of the current ventilation system have shown that in order to deliver the required air volumes to the production areas and throughout the mine, total intake airflow of 703 m\(^3\)/s is needed, of which 300 m\(^3\)/s is being delivered down the main fresh air raise (main FAR) and 103 m\(^3\)/s down the production shaft (Penna Shaft). The return airflow is then exhausted through the 6.7 m diameter main return air raise (Main RAR). In order to maintain minimum airflow velocities along major airways of the mined out areas, approximately 91 m\(^3\)/s is being transferred from the Main FAR to the “old operation” on level 122, which is then exhausted to the surface through the return air raises. This requires the mine’s main and booster fans to develop a total pressure of 10 kPa, with a combined fan power of 11,716 kW. Based upon electricity charges of $0.0242/kWh, the primary ventilation system’s operating cost is $2,400,000/year.

For the secondary distribution of airflow to the production areas, the mine uses various Joy\(^{TM}\), Woods\(^{TM}\) and Trimetal\(^{TM}\) auxiliary fans with motor powers ranging from 22 kW to 75 kW. In order to deliver the required air volumes into the production areas requires the mine’s auxiliary fans to develop a total power of 4,700 kW with an associated operating power cost of $934,500/year.

In addition to the primary and auxiliary ventilation operating power costs, the mine also spent another $710,000/year on natural gas to heat their intake air in the winter season and associated power costs of $340,000/year to provide cooling on levels 194, 200 and 215 during development and production processes on levels in the summer season. Combining electrical power for the primary ventilation, auxiliary ventilation and cooling systems with heating, the ventilation total...
operating cost is: $2,400,000/year (primary system) + $934,500/year (auxiliary system) + $710,000/year (heating) + $340,000/year (cooling) = 4.384M/year.

4. VENTILATION SYSTEM OPERATING COST FOR FUELCELL POWERED EQUIPMENT

In order to provide good working conditions during development and production processes assuming that fuelcell powered equipment will replace the primary diesels within the production areas, and based upon the mine’s production rate of 7,000 tons/day and a 90 cfm/tpd plus 25% additional airflow allowance for miscellaneous needs, the mine would require an intake airflow of 372 m³/s. However, in order to maintain minimum airflow velocities along the major airways and even throughout the inactive areas of the mine a total intake airflow of 605 m³/s is required, which represents a 14% reduction when compared to the diesel-based requirements.

To deliver and distribute this air volume to the active areas of the mine, ventilation simulations show that the required total pressure of the primary ventilation system drops to 7,400 Pa, with a combined surface and booster fan power of 7,500 kW and an associated power cost of $1,544,000/year, a 36% reduction.

For the auxiliary ventilation system, if the mine would downsize its current 75 kW production secondary fans to the next possible lowest size in its inventory, the ventilation system’s total fan power would drop to 4,060 kW. Based upon electricity charges of $0.0242/kWh, the auxiliary ventilation system’s operating power cost based on fuelcell powered primary production equipment has been estimated as $780,600/year, a 16% reduction.

As a consequence of a 14% reduction in total intake airflow, the heating costs during the winter season would similarly be reduced by 14% to $610,000/year. The operating power costs of the underground cooling system would also reduce to $292,000/year. In combination, to supply 605 m³/s to the working areas based upon airflow requirements determined according to fuelcell powered equipment, the combine energy operating cost would be: $1,544,000/year (primary system) + $780,600/year (secondary system) + $610,000/year (heating) + $292,000/year (cooling) = 3.226M/year, an overall 27% reduction.

5. ENVIRONMENTAL BENEFITS

For Laronde mine, a comparison of green house gas emissions (GHG) expressed in equivalent CO₂ emissions (in tonnes/year), for the current practice (using diesel powered equipment), versus fuelcells is given in Table 1. Upon replacing diesels with fuelcells, the combined ventilation electricity consumption, which includes the primary, auxiliary and cooling systems, reduces by 29%. The natural gas usage and diesel fuel consumption reduces by 14% and 75%, respectively. This assumes that within the fuelcell option approximately 25% of the remaining non-production small vehicles would still be powered by diesel engines. The conversion factors used were 200 tonnes of equivalent CO₂/GWh of electricity generated, 1.90 tonnes/ML of natural gas, and 2.777 tonnes/ML of diesel fuel.
Based upon the conversion factors, it can be seen that at this operation the fuelcell technology would reduce the mine’s attributable GHG emissions through reduced electricity, natural gas and diesel fuel consumption by 8,749 tonnes of equivalent CO₂/year, which represents a 29% reduction.

### Table 1 - CO₂ Equivalent GHG Emissions Reductions Upon the Adoption of Fuelcells

<table>
<thead>
<tr>
<th>Scenario</th>
<th>Airflow</th>
<th>Electricity</th>
<th>Natural Gas</th>
<th>Diesel Fuel</th>
<th>Total GHG</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Intake Airflow</td>
<td>Usage</td>
<td>CO₂ Equivalent Emissions</td>
<td>Usage</td>
<td>CO₂ Equivalent Emissions</td>
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<tr>
<td>Current Practice</td>
<td>703 m³/s</td>
<td>151,838 MWh</td>
<td>30,368 tonnes</td>
<td>5.08 ML</td>
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<td>Fuelcell Option</td>
<td>605 m³/s</td>
<td>108,124 MWh</td>
<td>21,625 tonnes</td>
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<td>GHG Reductions</td>
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</tbody>
</table>

6. **SUMMARY**

The analyses show potential for an overall 27% reduction in the combined electrical, heating and cooling costs of the primary and auxiliary ventilation system for associated average intake airflow reductions of 14%. Reductions in the electrical operating costs of the primary ventilation system are in the order of 36%. Reductions in the electrical operating cost of the production operating fans of the auxiliary ventilation system are in the order of 16%. Again this is due to their inflexibility to accommodate reduced airflow regimes during the mining cycle. For this operation, as a result of replacing diesels with fuelcells, heating and cooling cost reductions would be the same as the mine intake airflow reduction of approximately 14%.

These combined savings are less than those experienced within the primary system, because this mine needs to heat the intake airflow in winter season, provide cooling below level 194 during the summer season. Consequently, the auxiliary ventilation system cannot be reduced to the same degree as the primary ventilation system. On the other hand, similar to other underground operations within the Province of Quebec, some of the potential operating cost savings that could have materialized, have already been achieved due to the fact that at this mining operation, the amount of intake air volumes needed to dilute diesel pollutants have been determined based upon the “clean” diesel engine requirements such as 0.0475 m³/s per rated kW of diesel power for the “upper levels” and 0.053 m³/s/kW for the “lower levels”, which is approximately 25% and 15%, respectively, lower than the airflow requirements in the Province of Ontario, i.e. 0.063 m³/s per rated kW of diesel power. The comparative intake airflow and combined ventilation operating cost for fuelcells compared to diesel-powered equipment is given in Figure 2 and 3.
Figure 1 - Total Mine Intake Airflow – Diesels vs. Fuelcells

Figure 2 - Combined Ventilation Operating Cost – Diesels vs. Fuelcells
E. LOUVICOURT MINE - AUR RESOURCES LTD., QUEBEC, CANADA

1. INTRODUCTION

The Louvicourt copper, zinc mine and concentrator is located 20 km east of Val d’Or, Quebec. Its 4,300-tonne per day plant utilizes semi-autogenous grinding, froth flotation and pressure filters for recovery of metal concentrate. The mine has 290 employees, with an operation schedule of 5 x 8 hours/dayshift and 4x10 hours/night-shift for development operations and 10 hours shift (except Friday and Saturday night) for mucking. The orebody is 415-920 m below surface, accessed by two circular, concrete lined shafts, both 950 m deep. Mining entails large transverse blasthole stoping with backfill and subsequent pillar recovery. Typical stope dimensions are 30 m high, 15 m wide and 25 m long. Typical development dimensions are 5.5 m wide x 4.5 m high.

The layout of the mine is shown in the ventilation schematic, Figure 1. The mine has two main intake airways; Shaft #1 (the production shaft) from the surface down to the 950 m level, fed by two 75 hp fans in a parallel arrangement and Shaft #2 (exploration shaft) from the surface down to the 770 m level, which delivers airflow on the 475 m, 565 m, 655 m and 770 m levels by means of two 700 hp ABB fans also in a parallel arrangement.

![Figure 1 - Primary Ventilation System Schematic – Louvicourt Mine](image-url)
Air to the production areas is drawn off this intake system either naturally or under the assistance of booster fans. Auxiliary ducting then takes the air to the desired production locations. In the production areas, single-pass ventilation is employed. The return air is exhausted from the bottom of the mine to the 655 m level through an exhaust raise and the ramp system. From the 680 m level, the return air is then directed to the surface through an exhaust raise system connecting the 655 m and 475 m level, two incline airways and the main return air raise (Main RAR) by means of two 500 hp ABB fans also in parallel arrangement located at the top of the Main RAR.

2. CURRENT VENTILATION REQUIREMENTS AND DESIGN CRITERIA

In the current system, the controlling criterion for ventilation is diesel exhaust dilution. Secondary considerations are blast fume clearance and dust control. Due to its location (northern Quebec), and the relatively shallow/medium depth of the workings (between the surface and 950 m depth), heat is not currently an issue. Further to this due to the ore/strata configuration of the mine, there are no other strata gas issues such as methane that must be considered into the design criteria.

The airflow requirements for production and development operations are determined based upon the diesel exhaust criteria, which in the Province of Quebec acknowledge the use of clean diesel fuel and “clean” diesel engine technology. This mine uses 0.0475 m³/s per rated kW of diesel power in its overall primary design. This is approximately 25% lower than the air volume requirements in the province of Ontario of 0.063 m³/s per rated kW of diesel power.

3. VENTILATION SYSTEM OPERATING COST USING DIESEL POWERED EQUIPMENT

Ventilation modelling and simulation exercises of the current situation have shown that in order to deliver the required air volumes to the production areas and throughout the mine, intake airflow of 387 m³/s is needed during development and production processes. From the total intake airflow, 111 m³/s are delivered down the production shaft (Shaft #1) and 276 m³/s down the exploration shaft (Shaft #2). This requires the mines main and booster fans to develop a total pressure of 2.804 kPa, with a combined total power of 1447 kW. Based upon electricity charges of $0.387/kWh, the primary ventilation operating power cost is $419,400/year.

For the secondary distribution of airflow to the production areas, the mine uses approximately 82 auxiliary fans with motor powers ranging from 15 kW (20 hp) through to 112 kW (150 hp). The combined installed power of these fans is 4,370 kW. However, these auxiliary fans would only be operational according to need. Consequently, only 37%, sufficient for 8 active locations are operating during the mine’s development and production shifts. During statutory holidays all auxiliary ventilation systems are turned off. Therefore, the needed fan power for the secondary airflow distribution is 1,650 kW. Based upon this information and electricity charges of $0.0387/kWh, the auxiliary ventilation operating power cost in the current system have been estimated as $479,300/year. In addition to this, in 2002, the mine also spent another $800,000/year on natural gas to heat their intake air during the winter season.
Combining electrical power and heating, the ventilation system’s operating cost is: $419,400/year (primary system) + $479,300/year (auxiliary system) + $800,000/year (heating costs) = 1.699M/year.

4. VENTILATION SYSTEM OPERATING COST USING FUELCELL POWERED EQUIPMENT

Determining the appropriate ventilation requirements in a mine employing fuelcell powered equipment presents some level of complexity, depending upon the degree of conversion and on how the light-duty diesel vehicles are accounted for in the original design. However, assuming that the 4,300 tpd are produced from eight active stopes and using the 0.042 m³/s/tpd plus a 25% additional allowance for miscellaneous needs, and maintaining minimum flows within the inactive areas, the mine would require a total intake airflow of 318 m³/s, which represents a 18% reduction when compared to the diesel based requirement.

To deliver and distribute this airflow to the production areas, ventilation simulations show that the required total power of the primary ventilation system drops to 803 kW, with an associated operating cost of $232,600/year, a 45% reduction.

With regard to the auxiliary ventilation system, if the mine would downsize its current 112 kW and 75 kW “production” operations secondary fans to the next possible lowest size in its inventory, the auxiliary ventilation system’s total fan power would drop to 1320 kW. Based upon electricity charges of $0.0387/kWh, the auxiliary ventilation operating power cost based on fuelcell-powered equipment is estimated at $382,500/year, a 20% reduction.

As a consequence of a 18% reduction in total intake airflow, the heating costs during the winter season would similarly be reduced by 18% to $655,000/year. In combination to supply 318 m³/s to the working areas based upon airflow requirements determined according to fuelcell powered equipment, the combined energy operating cost would be: $232,600/year (primary system) + $382,500/year (auxiliary system) + $655,000/year (heating costs) = 1.27M/year, an overall 25% reduction. It should also be noted that from the combined energy operating cost of $1.27M/year, the primary and auxiliary fans account for 48.5% of the combined costs, heating accounts for 51.5%.

5. ENVIRONMENTAL BENEFITS

In Table 1, a comparison is given with regard to the greenhouse gas emissions (in equivalent CO₂ emissions, in tonnes/year) at the Louvicourt mine for diesels versus fuelcells, based upon their ventilation requirements as described in the previous sections. Upon replacing diesels with fuelcells, the combined ventilation electricity consumption at this operation reduces by 32%, the natural gas usage by 18% and the diesel consumption by approximately 75%. The conversion factors used were 200 tonnes of equivalent CO₂ emissions per GWh of electricity consumed, 1.90 tonnes/ML of natural gas consumed, and 2.777 tonnes/ML of diesel fuel consumed.
Based upon the conversion factors, it can be seen that at this operation the fuelcell technology would reduce the mine’s attributable GHG emissions by 1,469 tonnes of equivalent CO₂ emissions per year, which represents a 32% reduction.

Table 1 - CO₂ Equivalent GHG Emissions Reductions Upon the Adoption of Fuelcells

<table>
<thead>
<tr>
<th>Scenario</th>
<th>Airflow</th>
<th>Electricity</th>
<th>Natural Gas</th>
<th>Diesel Fuel</th>
<th>Total GHG</th>
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<td>Intake Airflow</td>
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<td>CO₂ Equivalent Emissions</td>
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<td>1.10</td>
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6. SUMMARY

The analyses show potential for an overall 25% reduction in the combined electrical and heating costs of the primary and auxiliary ventilation systems for associated average intake airflow reductions of 18%. Reductions in the electrical operating cost of the primary ventilation system are in the order of 45%, reductions in the electrical operating costs of the production operating fans of the auxiliary ventilation system may only be in the order of 20%. For this operation heating cost reductions would be the same as the average intake airflow reductions of approximately 18%.

These combined savings are somehow less than those experienced in the operating costs of the surface fans, because this mine needs to heat its intake airflow in the winter season, and again the auxiliary ventilation system may not be trimmed to the same degree as the primary ventilation system. Further to this, some of the potential operating cost savings that could have materialized, have already been achieved due to the fact that at this mining operation, the amount of air volumes needed to dilute diesel pollutants have been determined based upon “clean” diesel engine requirements such as 0.0475 m³/s per rated kW of diesel power. This is approximately 25% lower than the air volume requirements in the province of Ontario, i.e. 0.063 m³/s per rated kW. The comparative intake airflow and combined ventilation operating cost savings for fuelcells compared to diesel powered equipment is given in Figure 2 & 3.
Figure 2 - Total Mine Intake Airflow – Diesels vs. Fuelcells.

Figure 3 - Combined Ventilation Operating Costs – Diesels vs. Fuelcells
F. DOYON MINE – CAMBIOR LIMITED, QUEBEC, CANADA

1. INTRODUCTION

The Doyon Division is comprised of two adjacent underground mines, Doyon Mine and Mouska Mine. The Doyon Mine is situated on the Cadillac gold fault in the Abitibi region of Quebec, located approximately five kilometres east of the Mouska mine, near Pressiac, in Bousquet Township, 41 kilometres east of Rouyn-Noranda. The Doyon mine covers 1,451 hectares.

Exploration was first reported in Bousquet Township around 1910. In the 1960’s, more intense prospecting was carried out at the Doyon mine site by prospector Arthur Doyon, after whom the mine is named. In 1977, Long Lac Mineral Exploration Ltd. took over from Silver-Stack Mines Ltd. And a 120-hole drilling program was conducted prior to the deposit going into production. Open pit mining of Zones 1 and 2 began in 1980 and underground mining in 1986. In 1994, Barrick Gold Corporation took over Lac Minerals, and then in 1998, Cambior acquired Barrick’s 50% interest to become the sole owner of the Doyon Operation.

Following the acquisition of Barrick’s 50% interest in the Doyon mine, Cambior consolidated the Doyon and Mouska operations by creating the Doyon Davison. As a cost-cutting measure, the company began processing the Mouska ore at the Doyon milling facilities and shut down the Vezina mill, which had been processing the ore from Mouska mine.

For 2002, the Doyon Division produced 216,000 ounces of gold at a mine operating cost of $228 per ounce. The mill processed 1,287,000 tonnes of ore (3,500 tonnes/day) at an average grade of 5.5 g Au/t. Capital expenditures for the Doyon Division totalled $8.1 million in 2002, slightly higher than in 2001, due to an increase in exploration and development drilling and purchase of mine equipment.

Current proven mineral reserves at the Doyon Division stand at 8.0 million tonnes at a grade of 5.3 g Au/t, representing 1.4 million ounces contained. As a result of the deep drilling program in 2002, measured and indicated mineral resources increased to 1.9 million tonnes at a grade of 3.6 g Au/t and inferred mineral resources amounted to nearly 6.3 million tonnes at a grade of 5.2 g Au/t. For 2003 the production target at Doyon mine is 3,700 tonnes/day. The mill is expected to process 1.3 million tonnes at a grade of 5.4 g Au/t.

2. GEOLOGY

The Doyon deposit lies in a strongly altered and deformed corridor of the Blake river group. The deformation is characterized by east-west striking schistosity with 75 degrees dip towards the south. Most of the mineralization occurs as veins within Zone 2 and the West Zone. In Zone 1, mineralization occurs as irregular veinlets and in pyrite concentrations associated with the sericite-chlorite pyritized schist.
Vein type mineralization dominates. One of the main characteristics of the Doyon mine deposit is the variation in vein orientation. The veins adopt a cylindrical type distribution that appears fairly systematic. The main mineralized veins are in order of centimetres and composed of varying proportions of pyrite, quartz, chalcopyrite, carbonate and gold with smaller proportions of bornite, sericite and chlorite.

3. THE MINING PROCESS

3.1 The Mine Layout

Both a rectangular timbered Production Shaft and a Ramp system access the mine. The production shaft is 1,019 m deep and has 10 shaft stations, located approximately 60 meters apart (See Figure 1).

![Figure 1 - Doyon Mine Layout](image)

The underground crushing and loading facilities are located on two different levels, the crushers on levels 9 and 15 and the loading stations on level 10 and 16. The 4-meter by 4.5-meter (Wide) ramp system can access the main levels between the surface and Level 10. In addition, in the upper area of the mine (levels 2 to 8), the ramp splits in two, east and west in order to cover the
breadth of the mine. The access ramp is over 7 kilometres long. The main drifts, crosscuts, footwall drifts and miscellaneous access drifts range from 3.6 m x 3.6 m to 4.2 m x 4.2 m, depending on the size of the equipment required for the various production areas.

The gold is contained in narrow, high-grade veins that are mined as groups in sublevel blasthole stopes designed above the breakeven cutoff grade. Doyon mine employs several different mining methods depending on the type of rock in which the ore zones occur. For example, bulk mining is used in the “West Zone”, in the alaskite. A number of small narrow mineralized veins are combined into one large mining block. Individual mining of those particular veins would be uneconomical due to their grade and concentration.

3.2 Mining Methods

Most of the ore reserves at the Doyon mine are extracted using selective mining methods. The method is actually a “vertical retreat with longholes”. Where the ore is present on more than two levels or sublevels, the stopes are mined in an ascending pattern.

At the Doyon mine sublevel selective mining is also used, especially in areas where the ore veins are narrow but less continuous than for the vertical retreat method. The sublevel methods improve ore definition and consequently allow a more accurate stope design.

Finally, the “longitudinal retreat” method is used in the sericitic schist in Zone 1. As ground conditions are very poor in this zone, special attention is paid to development and mine sequencing.

3.3 The Haulage System

In each of the mining methods, once the ore has been mucked from the draw points, it is transported to the nearest ore pass. At Doyon all the ore passes connect to one of the three trolleys on levels 6, 8 or 12, with a combined tramming capacity of more than 1.5 million tonnes. These trolleys take the ore from the various ore passes located along the trolley drift and transport it to the main ore pass system near the production shaft. Depending on the level, the ore then goes to one of the two crushers. Once crushed the ore is loaded into the skips from one of the automatic loading stations and hoisted to the surface by the production hoist. The hoisting capacity of the production hoist is 1.7 million tonnes per year. On the surface the ore is then transported to the main ore bin, then to the mill.

4. THE VENTILATION SYSTEM

The mine has three main intake airways; No.1 fresh air raise, from surface down to level 10-2 (600 m depth), fed by a 1.8 m diameter (72”), 250 hp surface fan, No.2 fresh air raise from surface down to level 6-0 (380 m depth) fed by two 1.8 m (72”) diameter main fans in parallel arrangement and No. 3 fresh air raise from surface down to level 8-0 (500 m depth) fed by two
2.1m (84”) diameter, 300 hp main fans also in a parallel arrangement. No.1 and No.3 fresh air raises deliver airflow to level 10-2 and level 8-0 respectively, and through the 3m diameter internal raise down to level 12-0. No.2 fresh air raise delivers airflow to level 6-0 and through an internal raise system to the bottom of the mine, level 16-0 (1000 m depth) as shown in Figure 2.

![Primary Ventilation System Schematic – Doyon Mine](image)

**Figure 2 - Doyon Mine Primary ventilation System Schematic**

Fresh air to the production areas is drawn off of this intake system either naturally or under the assistance of booster fans. In the production areas series auxiliary ventilation system is employed. The return air is then exhausted from the bottom of the mine (level 16-0) through the production shaft under the assistance of three booster fans located on levels 14-0, 15-0 and 16-0 and two main ramp systems.

5. **CURRENT VENTILATION REQUIREMENTS**

In the current system, the controlling criterion for ventilation is diesel exhaust dilution. Secondary considerations are blast fume clearance and dust control. Due to the medium depth of the workings (between surface and 1000 m depth), currently heat is not an issue. There are no other strata gas issues (i.e. methane) that must be considered within the ventilation design criteria.
The airflow requirements for development and production processes are determined based upon the use of “clean” diesel engine technology. This mine uses 0.0475 m$^3$/s per rated kW of diesel power in its overall primary ventilation design, which is approximately 25% less than the air volume requirements in the province of Ontario of 0.063 m$^3$/s per rated kW.

6. VENTILATION SYSTEM OPERATION COST USING DIESELS

Under the current mining conditions, to deliver the required air volumes to the production areas and maintain airflow velocities within the inactive areas above 0.5 m/s, total intake airflow of 354 m$^3$/s is needed. From this, 76 m$^3$/s are delivered down the No.1 fresh air raise (No.1 FAR), 165 m$^3$/s through the No.2 fresh air raise (No.3 FAR) and 113 m$^3$/s through the newly constructed No.3 FAR. This requires the mine’s main and booster fans to develop a total pressure of 4,240 Pa, with a total power of 1,530 kW, and based upon electricity charges of $0.042/kWh, the primary ventilation operating power cost has been estimated at $550,000/year.

As mentioned before, most of the ore reserves at the Doyon mine are extracted using selective mining methods. The stopes usually follow the high-grade narrow veins and they are located within “Zone 1”, “Zone 2” and “West Zone” mining areas, relatively far from the vertical fresh intake airways. To overcome the pressure losses along the auxiliary ducting system, the mine uses approximately 226 auxiliary fans with motor powers ranging from 15 kW through to 120 kW with a total installed power of 12,660 kW. However, these auxiliary fans are only operational when needed, therefore, only 40% sufficient for 22 active locations are operating during the mine’s development and production shifts. During shift change and statutory holidays all auxiliary ventilation fans are turned off. As a result, the needed fan power for the secondary airflow distribution is approximately 5,064 kW. Based upon electricity charges of $0.0420/kWh, the auxiliary ventilation operating power cost is has been estimated at $1,635,000/year.

In addition to this, in 2002, the mine also spent another $320,000/year on natural gas to heat its intake air during the winter season.

Combining electrical power and heating Doyon mine’s ventilation system operating cost under current mining conditions is: $550,000/year (primary system) + $1,635,000/year (secondary system) + $320,000/year (heating costs) = $2.505M/year.

7. VENTILATION SYSTEM OPERATING COST USING FUELCELLS

Assuming that 4,000 tpd are produced from 22 active stopes and using the 90 cfm/tpd (0.042 m$^3$/tpd) plus a 25% additional allowance for miscellaneous needs and maintaining minimum air velocities within the inactive areas, the mine would require total intake airflow of 276 m$^3$/s, which represents a 22% reduction when compared to diesel based equipment.

To deliver and distribute this airflow to the production areas, the required total power of the primary ventilation system drops to 723 kW, with an associated operating cost of $260,000/year, a 53% reduction.
Regarding the auxiliary ventilation system, if the mine would down-size its current 120 kW, 110 kW and 90 kW “production” operations secondary fans to the next possible lowest size in its inventory (i.e. 90 kW, 75 kW and 56 kW, respectively), the auxiliary ventilation system’s total fan power would drop to 4,118 kW. Based upon electricity charges of $0.0420/kWh, the auxiliary ventilation operating power cost based on fuelcell-powered equipment has been estimated at $1,330,000/year, a 18% reduction.

As a consequence of a 22% reduction on total intake airflow, the heating costs during the winter season would similarly be reduced by 22% to $250,000/year. In combination, to supply 318 m³/s to the working areas based upon airflow requirements determined according to fuelcell powered equipment, the combined energy operating cost would be: $260,000/year (primary system) + $250,000/year (secondary system) + $250,000/year (heating costs) = 1.840M/year, an overall 26.5% reduction. It should also be noted that from the combined energy operating cost of $1.840M/year, the primary fans account for 14%, the auxiliary fans for 72%.

### 8. ENVIRONMENTAL BENEFITS

**Table 1 - CO₂ Equivalent GHG Emissions Reductions Upon the Adoption of Fuelcells**

<table>
<thead>
<tr>
<th>Scenario</th>
<th>Airflow</th>
<th>Electricity</th>
<th>Natural Gas</th>
<th>Diesel Fuel</th>
<th>Total GHG</th>
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<tbody>
<tr>
<td></td>
<td>Intake Airflow</td>
<td>Usage</td>
<td>CO₂ Equivalent Emissions</td>
<td>Usage</td>
<td>CO₂ Equivalent Emissions</td>
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<td></td>
<td>m³/s</td>
<td>MWh</td>
<td>tonnes</td>
<td>ML</td>
<td>tonnes</td>
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<td>Current Practice</td>
<td>354</td>
<td>52,023</td>
<td>10,402</td>
<td>2.90</td>
<td>2.49</td>
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<tr>
<td>Fuelcell Option</td>
<td>276</td>
<td>37,857</td>
<td>7,572</td>
<td>2.27</td>
<td>1.94</td>
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<tr>
<td>GHG Reductions</td>
<td></td>
<td>2.88</td>
<td>0.55</td>
<td>2.34</td>
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</tbody>
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In Table 1, a comparison is given of the equivalent CO₂ emissions (tonnes/year) for the current operation and the adoption of the fuelcell technology, based upon their ventilation requirements as previously described. Upon replacing diesels with fuelcells, the combined ventilation electricity consumption at Doyon mine reduces by 27.3%, the natural gas usage by 22% and the diesel consumption by approximately 75%. The conversion factors used were 200 tonnes/GWh of electricity consumed, 1.90 tonnes/ML of natural gas, and 2.777 tonnes/ML of diesel fuel consumed.

Based upon the conversion factors, it can be seen that at this operation the fuelcell technology would reduce the mine’s attributable GHG emissions by 2,833 tonnes/year, which represents a 27% reduction. This can depend upon any energy savings from hydrogen generation.
9. SUMMARY

This analysis shows potential for an overall 26.5% reduction in the combined electrical and heating costs of the primary and auxiliary ventilation systems for associated average intake airflow reductions of 22%. Reductions in the electrical operating cost of the primary ventilation system are in the order of 53%, reductions in the electrical operating cost of the auxiliary ventilation system may only be in the order of 18%. Upon replacing diesels with fuelcells, heating cost reductions would be the same as the average intake airflow reductions of approximately 22%.

The comparative intake airflow and combined ventilation operating cost savings for fuelcells compared to diesel powered equipment is given in Figure 3 and Figure 4.
Figure 3 - Total Mine Intake Airflow – Diesels vs. Fuelcells

Figure 4 - Combined Ventilation Operating Cost – Diesels vs. Fuelcells
## APPENDIX 2 – CLASSIFICATION OF CANADIAN MINING OPERATIONS

### BASE METAL MINES

<table>
<thead>
<tr>
<th>Location</th>
<th>Minerals recovered</th>
<th>Ore mined</th>
<th>Ore type</th>
<th>Mining method</th>
<th>Depth</th>
<th>LHs</th>
<th>Loaders</th>
<th>Trucks</th>
<th>Main fans</th>
<th>Diameter (ft)</th>
<th>Capacity (tph)</th>
<th>Brake hp</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>INCO</strong> Collumar/Old Red East Copper Cliff, ON P0M 1N0 Tel: 705-665-4340</td>
<td>Ni, Cu, Co, Au Pt, Pd, Ag</td>
<td>3775 tpd (3426mtpd)</td>
<td>massive sulphides with high grade Cu stringers</td>
<td>bulk Vertical Reniat</td>
<td>3400</td>
<td>4100</td>
<td>Wagner STBB, Elphinstone R1790 MTI Wagner 2 Joy 2 Joy 2 Joy 2 Joy 2 Joy 2 Joy</td>
<td>72</td>
<td>150000</td>
<td>180</td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>INCO</strong> Copper Cliff South Mine Copper Cliff, ON P0R 1N0 Tel: 705-662-6021</td>
<td>Ni, Cu, Co, Au Pt, Pd, Ag</td>
<td>3000 Tpd (2725mtpd)</td>
<td>massive sulphide and disseminated quartz veins</td>
<td>Vertical Reniat Blasting upper Blasting</td>
<td>4260</td>
<td>3500</td>
<td>Wagner STBBs JS426</td>
<td>84</td>
<td>160000</td>
<td>190</td>
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<tr>
<td><strong>INCO</strong> Crean Hill Mine Copper Cliff, ON P0M 1N0 Tel: 705-665-7270</td>
<td>Ni, Cu, Co, Au Pt, Pd, Ag</td>
<td>1500 tpd (1330mtpd)</td>
<td>disseminated sulphide</td>
<td>Bulk VCR Bulk UCR</td>
<td>4100</td>
<td>Wagner ESTBA Elphinstone 8 yd diesel Wagner EST 8A</td>
<td>84</td>
<td>150000</td>
<td>180</td>
<td></td>
<td></td>
<td></td>
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<tr>
<td><strong>INCO</strong> Creighton Mine Copper Cliff, ON P0M 1N0 Tel: 705-663-6008</td>
<td>Ni, Cu, Co, Au Pt, Pd, Ag</td>
<td>4000 tpd (3840mtpd) 303 dtyr</td>
<td>disseminated sulphide in nickel to metakal sulphide</td>
<td>vertical Reniat</td>
<td>2211</td>
<td>7137</td>
<td>Wagner 60t Wagner 60c Krom 55t</td>
<td>4 Joy 2 Joy</td>
<td>225000</td>
<td>450</td>
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<td></td>
</tr>
<tr>
<td><strong>Inco</strong> Garson mine Copper Cliff, ON P0M 1N0 Tel: 705-663-0008</td>
<td>Ni, Cu, Co, Au Pt, Pd, Ag</td>
<td>2000 tpd (1100mtpd)</td>
<td>massive sulphide</td>
<td>below 906ft Blasting above 356ft Blasting</td>
<td>4241</td>
<td>4 Wagner STBB 1 Wagner 7 yd2 1 Wagner 3 3 yd2 1 JCS 2 yd2 1 JCI 1 5/ yards</td>
<td>5 JCS 400</td>
<td>90</td>
<td>225000</td>
<td>450</td>
<td></td>
<td></td>
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<tr>
<td><strong>Inco</strong> Garson mine Copper Cliff, ON P0M 1N0 Tel: 705-663-0008</td>
<td>Ni, Cu, Co, Au Pt, Pd, Ag</td>
<td>3600 mtpd (3000dp)</td>
<td>massive and disseminated sulphides</td>
<td>Blasting Cut &amp; Fill With ramp access &amp; post pillar</td>
<td>1465m</td>
<td>2 Wagner STBB 2 Wagner 7 7/ yards 4 ton 600</td>
<td>1 JCI, 26 t 1 Wagner 37 t</td>
<td>90</td>
<td>225000</td>
<td>450</td>
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<td></td>
</tr>
<tr>
<td><strong>Falconbridge</strong> Craig Mine Ongina, ON P0M 1N0 Tel: 705-663-2761</td>
<td>Ni, Cu, Co, Au Pt, Pd, Ag</td>
<td>2100 mtpd (2300dp)</td>
<td>massive sulphides</td>
<td>Cut &amp; Fill Longhole uppers &amp; Blasting</td>
<td>5200</td>
<td>3 LHs 9 yd2 3 LHs 9 yd2</td>
<td>1 Tom 40t 1 Wagner 37 t</td>
<td>90</td>
<td>350000</td>
<td>700</td>
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<tr>
<td><strong>Falconbridge</strong> Octave Mine Ongina, ON P0M 1N0 Tel: 705-663-3411</td>
<td>Ni, Cu, Co, Au Pt, Pd, Ag</td>
<td>11000 tpd (10000mtpd)</td>
<td>massive sulphides</td>
<td>Cut &amp; Fill (36%) Longhole upper-64%</td>
<td>12600m 12300m</td>
<td>4 Wagner STBB 62 2 JCS 5600</td>
<td>1 JCI 26 t 2 JCS 5600</td>
<td>1 Wagner 26 t</td>
<td>2 TLT Babcock 2 TLT Babcock 2 TLT Babcock</td>
<td>84</td>
<td>240000</td>
<td>250</td>
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<tr>
<td><strong>Falconbridge</strong> Sorensens Mine Ongina, ON P0M 1N0 Tel: 705-663-3411</td>
<td>Ni, Cu, Co, Au Pt, Pd, Ag</td>
<td>1300 mtpd (1040dp)</td>
<td>massive sulphide veins</td>
<td>mechanized Cut &amp; Fill narrow veins Longhole</td>
<td>3100</td>
<td>5000</td>
<td>Wagner STBB</td>
<td>1 Wagner 26 t</td>
<td>2 TLT Babcock 2 TLT Babcock 2 TLT Babcock 2 TLT Babcock 2 TLT Babcock 1 Applier</td>
<td>54</td>
<td>200000</td>
<td>1200</td>
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<tr>
<td><strong>Falconbridge</strong> Southills Mine Ongina, ON P0M 1N0 Tel: 705-663-3411</td>
<td>Ni, Cu, Co, Au Pt, Pd, Ag</td>
<td>1100 mtpd (1400dp)</td>
<td>massive and disseminated sulphides</td>
<td>post pillar Cut &amp; Fill Longhole</td>
<td>1638m</td>
<td>3 Wagner STBB</td>
<td>1 Wagner 26 t</td>
<td>2 TLT Babcock 2 TLT Babcock 2 TLT Babcock 2 TLT Babcock 2 TLT Babcock 2 TLT Babcock</td>
<td>54</td>
<td>200000</td>
<td>1200</td>
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<tr>
<th>Location</th>
<th>Minerals recovered</th>
<th>Ore mined</th>
<th>Ore type</th>
<th>Location</th>
<th>Mining method</th>
<th>Depth (m)</th>
<th>Diameters (mm)</th>
<th>Capacity (t/hr)</th>
<th>Capacity (t/hr)</th>
<th>Diameters (mm)</th>
<th>Capabilities</th>
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<td>Breakwater Resources, Ltd</td>
<td>Zn, Cu, Ag</td>
<td>3060 mtsp</td>
<td>Massive sulphides</td>
<td>Transverse Blasthole</td>
<td>747m</td>
<td>4 Tera 4610</td>
<td>2 JC 37426</td>
<td>2.1 m</td>
<td>40,000 t</td>
<td>2.1 m</td>
<td>450</td>
</tr>
<tr>
<td>Breakwater Resources, Ltd</td>
<td>Zn, Cu</td>
<td>1000 mtsp</td>
<td>Volcanogenic massive sulphide</td>
<td>Longhole 4 1/2</td>
<td>900m</td>
<td>5 Tera 1300, 5 yd</td>
<td>1 Ton 400 elec, 5 yd</td>
<td>4 Tera 4610</td>
<td>1.4 m</td>
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<td>2000 mtsp</td>
<td>Banded massive sulphide</td>
<td>Stalactite, benching, room &amp; pillar</td>
<td>2 JC 3580</td>
<td>1 Ton 9 yd</td>
<td>1 Tera 6 yd</td>
<td>2 Cat 980C</td>
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<td>7500 t</td>
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<td>Hudson Bay Mining and Smelting Co Ltd</td>
<td>Zn, Cu, Ag</td>
<td>1500 mtsp</td>
<td>Structurally stacked massive sulphides</td>
<td>Longhole Retreat Mechanized drilling</td>
<td>795m</td>
<td>1 Tera 8 yd</td>
<td>3 Tera 10 yd</td>
<td>3 Wagner 316</td>
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<td>12,000 t</td>
<td>250</td>
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<td>Zn, Cu, Ag</td>
<td>800 mtsp</td>
<td>Massive to disseminated sulphides</td>
<td>Mechanized Cut &amp; Fill</td>
<td>3 Wagner 1775</td>
<td>4 Wagner 1444</td>
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<td>9000 t</td>
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<td>Hudson Bay Mining and Smelting Co Ltd</td>
<td>Zn, Cu, Ag</td>
<td>5000 mtsp</td>
<td>Sulfide enrichive massive sulphide &amp; pyrite</td>
<td>Blast Hole With Blast Cap</td>
<td>2000</td>
<td>Tera 1400, 8 yd</td>
<td>3 Wagner 430</td>
<td>7.0 yd</td>
<td>27,000 t</td>
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<td>Noranda Inc.</td>
<td>Pb, Zn, Cu, Ag</td>
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<td>14 Tera 10 yd</td>
<td>24 Wagner STBB</td>
<td>14 Wagner 1433</td>
<td>9.0 yd</td>
<td>30,000 t</td>
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<td>Massive to disseminated sulphides</td>
<td>Longhole mechanized</td>
<td>1100m</td>
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<td>3 Tera 15 yd</td>
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<td>2 Tera 500</td>
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<td>Sedimentary enrichive massive sulphide</td>
<td>Room &amp; pillar</td>
<td>3.6 Tera 420</td>
<td>1 Tera 8 yd</td>
<td>2 Wagner 725</td>
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<td>10000 t</td>
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<td>Aerolocourt Inc., Ltd</td>
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<td>946m</td>
<td>1 Tera 756</td>
<td>8 Tera 10 yd</td>
<td>2 Tera 300</td>
<td>2 Tera 500</td>
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### PRECIOUS METAL MINES

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<tr>
<th>Location</th>
<th>Minerals recovered</th>
<th>Ore mined</th>
<th>Ore type</th>
<th>Mining method</th>
<th>Depth</th>
<th>LHDs</th>
<th>Loaders</th>
<th>Trucks</th>
<th>Main fans</th>
<th>Diameter (in)</th>
<th>Capacity (MT)</th>
<th>Drive hp</th>
</tr>
</thead>
<tbody>
<tr>
<td>Barrick Gold Corp</td>
<td>Au, Ag, Cu</td>
<td>2200 mtpd (46480 tpd)</td>
<td>Volcaniclastic Main</td>
<td>Stoping with real-fill and cemented rockfill</td>
<td>1244 m</td>
<td>Wagner STB 1000 Ton 4500</td>
<td>Tamrock Tere 401</td>
<td>2 Fans 60-50</td>
<td>98.4 ft</td>
<td>7000</td>
<td>200</td>
<td></td>
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<tr>
<td>Earlitz-Oriental Mine</td>
<td>Au, Ag, Cu</td>
<td>2151 mtpd (43711 tpd)</td>
<td>finely disseminated</td>
<td>Longhole open</td>
<td>1300 m</td>
<td>Wagner 2 yd³ Elec. &amp; diesel</td>
<td>1 Canadian Borer 75</td>
<td>1 Canadian Borer 66</td>
<td>1 Canadian Borer 54</td>
<td>1 Canadian Borer 49</td>
<td>20000</td>
<td>200</td>
</tr>
<tr>
<td>Battle Mountain Canada Ltd</td>
<td>Au, Ag, Cu</td>
<td>3000 mtpd (63000 tpd)</td>
<td>disseminated gold in potassic and schist altered</td>
<td>Blockhole</td>
<td>1146 m</td>
<td>9 JC JOHNS, 6 yd³ 2 Wagner, 8 yd³ 3 Elphinstone, 8 yd³ 5 Tamrock, 2 yd³ 2 Fans 78-30</td>
<td>2 Fans 78-30</td>
<td>2 Fans 54-25</td>
<td>1 Canadian Borer 75</td>
<td>1 Canadian Borer 66</td>
<td>20000</td>
<td>200</td>
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<tr>
<td>Battle Mountain Canada Ltd</td>
<td>Au, Ag, Cu</td>
<td>2000 mtpd (40000 tpd)</td>
<td>disseminated pyrite in phyllite altered</td>
<td>Sublonghole</td>
<td>967 m</td>
<td>5 MT LHDs, 7 yd³ 3-Capac LHDs, 3.5 yd³</td>
<td>2 Fans 78-30</td>
<td>2 Fans 54-25</td>
<td>1 Canadian Borer 75</td>
<td>1 Canadian Borer 66</td>
<td>20000</td>
<td>200</td>
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<tr>
<td>Centurion Inc</td>
<td>Au, Ag, Cu</td>
<td>3500 mtpd</td>
<td>No 1 zone: Pyrite and Quartz vein No 2, central &amp; west zones</td>
<td>Transverse Longhole</td>
<td>1000 m</td>
<td>2 Fans 78-30</td>
<td>2 Fans 78-30</td>
<td>2 Fans 54-25</td>
<td>1 Canadian Borer 75</td>
<td>1 Canadian Borer 66</td>
<td>20000</td>
<td>200</td>
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<tr>
<td>Miramar Con Mine Ltd</td>
<td>Au, Ag, Cu</td>
<td>500 tpd (990 tpd)</td>
<td>shear hoisted</td>
<td>Shear hoist</td>
<td>940 m</td>
<td>2 Woods 175B RPM</td>
<td>2 Fans 78-30</td>
<td>2 Fans 54-25</td>
<td>1 Canadian Borer 75</td>
<td>1 Canadian Borer 66</td>
<td>20000</td>
<td>200</td>
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<tr>
<td>Miramar Giant Ltd</td>
<td>Au, Ag, Cu</td>
<td>300 tpd (570 tpd)</td>
<td>scheelite</td>
<td>Centrifuge</td>
<td>750 m</td>
<td>2 Fans 78-30</td>
<td>2 Fans 78-30</td>
<td>2 Fans 54-25</td>
<td>1 Canadian Borer 75</td>
<td>1 Canadian Borer 66</td>
<td>20000</td>
<td>200</td>
</tr>
<tr>
<td>Place Dene Canada</td>
<td>Au, Ag, Cu</td>
<td>1600 tpd (3140 tpd)</td>
<td>quartz carbonate veins in pyrite and pyrobitite With native gold, arsenopyrite, stibnite, chalcopyrite</td>
<td>Longhole Cut &amp; Fill</td>
<td>4312 m</td>
<td>4 Fans 1.5 yd³, 7 Fans 2 yd³ 2 Fans 5 yd³ 3 Wager, 16-1</td>
<td>1 Canadian Borer 75</td>
<td>1 Canadian Borer 66</td>
<td>1 Canadian Borer 54</td>
<td>20000</td>
<td>200</td>
<td></td>
</tr>
<tr>
<td>Place Dene Canada</td>
<td>Au, Ag, Cu</td>
<td>3000 tpd (6000 tpd)</td>
<td>quartz veins and stockworks</td>
<td>Narrow vein Cut &amp; Fill</td>
<td>5472 m</td>
<td>6 Fans 650</td>
<td>6 Fans 650</td>
<td>6 Fans 650</td>
<td>1 Canadian Borer 75</td>
<td>1 Canadian Borer 66</td>
<td>20000</td>
<td>200</td>
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<td>Place Dene Canada</td>
<td>Au, Ag, Cu</td>
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<td>quartz veins and stockworks</td>
<td>Narrow vein Cut &amp; Fill</td>
<td>5472 m</td>
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<td>6 Fans 650</td>
<td>1 Canadian Borer 75</td>
<td>1 Canadian Borer 66</td>
<td>20000</td>
<td>200</td>
</tr>
<tr>
<td>Place Dene Canada</td>
<td>Au, Ag, Cu</td>
<td>3000 tpd (6000 tpd)</td>
<td>quartz veins and stockworks</td>
<td>Longhole transverse</td>
<td>240m</td>
<td>5 Fans 650</td>
<td>5 Fans 650</td>
<td>5 Fans 650</td>
<td>1 Canadian Borer 75</td>
<td>1 Canadian Borer 66</td>
<td>20000</td>
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<td>3000 tpd (6000 tpd)</td>
<td>quartz veins and stockworks</td>
<td>Longhole transverse</td>
<td>240m</td>
<td>5 Fans 650</td>
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<td>5 Fans 650</td>
<td>1 Canadian Borer 75</td>
<td>1 Canadian Borer 66</td>
<td>20000</td>
<td>200</td>
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<tr>
<td>Place Dene Canada</td>
<td>Au, Ag, Cu</td>
<td>3000 tpd (6000 tpd)</td>
<td>quartz veins and stockworks</td>
<td>Longhole transverse</td>
<td>240m</td>
<td>5 Fans 650</td>
<td>5 Fans 650</td>
<td>5 Fans 650</td>
<td>1 Canadian Borer 75</td>
<td>1 Canadian Borer 66</td>
<td>20000</td>
<td>200</td>
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</table>
## Precious Metal Mines

<table>
<thead>
<tr>
<th>Location</th>
<th>Minerals Recovered</th>
<th>Ore Type</th>
<th>Mineralization</th>
<th>Mining Method</th>
<th>Depth</th>
<th>LHDs</th>
<th>Loaders</th>
<th>Trucks</th>
<th>Main Fans</th>
<th>Diameter (in)</th>
<th>Capacity (CFM)</th>
<th>Brake HP</th>
</tr>
</thead>
<tbody>
<tr>
<td>Richmond Mines Ltd.</td>
<td>Au, Cu, Ag</td>
<td>gold in disseminated arsenopyrite and pyrite</td>
<td>Longhole Undercut</td>
<td>Wagner STS 5</td>
<td>2000 ft</td>
<td>20 EJC 07450</td>
<td>Wagner M16</td>
<td>unspecified</td>
<td>72</td>
<td>47000</td>
<td>125</td>
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<tr>
<td>Tyna Gold, Inc. New Idar Mine</td>
<td>Au, Ag</td>
<td>gold in disseminated arsenopyrite and pyrite</td>
<td>Longhole Undercut</td>
<td>Wagner STS 5</td>
<td>1500 ft</td>
<td>20 EJC 07450</td>
<td>Wagner M16</td>
<td>unspecified</td>
<td>93</td>
<td>150000</td>
<td>125</td>
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<td>St. Anthony Mine</td>
<td>Au, Cu, Ag</td>
<td>gold in disseminated arsenopyrite and pyrite</td>
<td>Longhole Undercut</td>
<td>Wagner STS 5</td>
<td>2000 ft</td>
<td>20 EJC 07450</td>
<td>Wagner M16</td>
<td>unspecified</td>
<td>72</td>
<td>37000</td>
<td>100</td>
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<tr>
<td>Neston Resources, Inc. Joe Main Mine</td>
<td>Au, Cu, Ag</td>
<td>gold in disseminated arsenopyrite and pyrite</td>
<td>Longhole Undercut</td>
<td>Wagner STS 5</td>
<td>1500 ft</td>
<td>20 EJC 07450</td>
<td>Wagner M16</td>
<td>unspecified</td>
<td>93</td>
<td>150000</td>
<td>125</td>
<td></td>
</tr>
<tr>
<td>Alamos-Eagle, Lacolle Mine</td>
<td>Au, Ag, Cu, Zn</td>
<td>gold in disseminated arsenopyrite and pyrite</td>
<td>Longhole Undercut</td>
<td>Wagner STS 5</td>
<td>1500 ft</td>
<td>20 EJC 07450</td>
<td>Wagner M16</td>
<td>unspecified</td>
<td>72</td>
<td>37000</td>
<td>100</td>
<td></td>
</tr>
<tr>
<td>Goldcorp, Inc.</td>
<td>Au, Ag</td>
<td>gold in disseminated arsenopyrite and pyrite</td>
<td>Longhole Undercut</td>
<td>Wagner STS 5</td>
<td>1500 ft</td>
<td>20 EJC 07450</td>
<td>Wagner M16</td>
<td>unspecified</td>
<td>72</td>
<td>37000</td>
<td>100</td>
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</tbody>
</table>

## Metal Mines - Uranium

<table>
<thead>
<tr>
<th>Location</th>
<th>Minerals Recovered</th>
<th>Ore Type</th>
<th>Mining Method</th>
<th>Depth</th>
<th>LHDs</th>
<th>Loaders</th>
<th>Trucks</th>
<th>Main Fans</th>
<th>Diameter (in)</th>
<th>Capacity (CFM)</th>
<th>Brake HP</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cameco Resources</td>
<td>U</td>
<td>high grade lentic</td>
<td>Undercut &amp; Fill</td>
<td>160m</td>
<td>160m</td>
<td>B-JC JS220</td>
<td>B-JC J07415</td>
<td>Cameco</td>
<td>84</td>
<td>100 m/sec</td>
<td>250</td>
</tr>
</tbody>
</table>

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