**ALPHA FOUNDATION FOR THE IMPROVEMENT OF MINE SAFETY AND HEALTH**

# Final Technical Report

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| **Project Title:** | Free and Open Source Professional Development for the Mine Ventilation Community: An Innovative Approach to Improving Competence |
| **Grant Number:** | AFC-215-42 |
| **Organization:** | Virginia Tech |
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| **Period of Performance:** | July 1, 2015 to June 30, 2017 |

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# 2.0 Executive Summary

Catastrophic failures of engineered ventilation controls, or a failure to understand the role of the mine environment on the ventilation system have resulted in multiple fatality incidents many times over the years; most recently, the Sago Mine and Upper Big Branch Mine Explosions highlight the need for improved education and professional development resources. The group of mining professionals with expert competence in ventilation is small and dwindling, despite the fact that most mine engineers will, at some point, be involved in ventilation engineering. For professionals who want to improve their competence there are relatively few opportunities for professional development and even fewer specialized consultants. This project aims to develop an innovative set of online tools for underground coal, metal, and non-metal ventilation engineering training with considerable impact on safety in underground mines. This work has involved investigating and collating best practice at state-of-the-art underground mines around the world, and distributing this knowledge via an innovative online course platform. Based on the premise that knowledge should be accessible, the course has been distributed via a free and open source platform, *Canvas Instructure* (Canvas, 2017). It has also been briefly piloted during the project, and will be maintained with researchers and professionals adding material for community consumption. To get an invitation and access the course online go to https://sites.google.com/view/aeoluslaunch/get-an-invite-to-aeolus.

A total of 12 courses have been developed in Mine Ventilation including: , Introduction to Mine Ventilation, Automation, Bulk Air Heating and Cooling, Diesel Particulate Matter, Dust, Fans, Gaseous Contaminants, Modeling and Simulation, Underground Coal Ventilation Design, Underground Metal/Non-metal Ventilation Design, Ventilation Economics, and Ventilation Surveys. Each course has a prescribed plan of study, but users may also access various topics via an index. A stand alone introductory site has also been developed where users can request an invitation to the courses, submit additional information for peer review or suggestions for improvement. The invitations are necessary because the Canvas platform requires that unique users be enrolled, which also allows the user to track their own progress, and engage in discussion. Going forward, this will also allow us to track enrollment trends. For all intents and purposes, the platform is entirely open to any user.

The courses have been made available to expert reviewers and further expert reviews will be requested. This approach has also been discussed at length with ventilation engineers around the world as we have collected case studies, the Underground Ventilation Committee of the SME, and practitioners at the 2017 North American Mine Ventilation Symposium; it has been met with positive support.

# 3.0 Problem Statement and Objectives

The group of mining professionals with expert competence in ventilation is small and dwindling, despite the fact that most mine engineers will, at some point, be involved in ventilation engineering. For professionals who want to improve their competence there are relatively few opportunities for professional development and even fewer specialized consultants. This proposal aims to develop an innovative set of online tools for underground coal and metal/non-metal ventilation engineering training with considerable impact on safety in underground mines. This work will involve investigating and collating best practice at state-of-the-art underground mines around the world, and distributing this knowledge via an innovative online course platform.

A brief discussion of the development of this idea is instructive. Prior to 2015, Pierre Mousset-Jones, Professor at the University of Nevada-Reno, maintained an email listserve of ventilation professionals where he regularly disseminated information and stoked discussion. He had approximately 300 users from around the world, and spent a great deal of time on this. His work was regularly acknowledged as an enormous professional service to the ventilation community. Upon his retirement, the UVC, after much discussion, moved the group to an SME Community and requested an exception from SME to allow non-SME members to join, which was granted. Nonetheless, much of the discussion among these groups has been how to improve competence. Everything from university curricula to requiring certified ventilation officers in the US (e.g., the Australian system) has been discussed. Ultimately, the group agreed that some sort of repository was needed, but did not solve the problem of resources to initiate the design and population of such a platform. Hence, this proposal was borne of these discussions.

Based on the premise that knowledge should be accessible, these courses are distributed via an open platform, *Canvas Instructure*. The primary objectives of this work are to i) collect and distribute best practice and basic and advanced principles of mine ventilation engineering for the improved competency of the industry; ii) to demonstrate an innovative online platform for professional development in the mining industry; and iii) to enhance immediate mine safety while building a body of knowledge that will continuously enhance safety.

The materials developed are targeted toward the US industry, and, although international case studies are utilized they are described in the context of US regulation, with beyond compliance controls and leading practices identified. This work is particularly novel because few free and open source materials are available to mining professionals, and none through a formal learning management system.

The specific aim of this work was to develop a universally accessible professional development platform to enhance safety in underground mines. This work allows for the dissemination of knowledge, including theory, best practice, and practical application in mine ventilation engineering, and the platform is "living" – we expect to update and revise it continuously. Ultimately, the work is targeted toward practitioners with engineering backgrounds, but it is also appropriate for workers engaged in mine ventilation work without any such formal training (e.g., foremen and supervisors). By providing unlimited access to professional development for ventilation professionals, catastrophic and routine failures of engineered ventilation systems may be avoided, with a considerable positive impact on underground safety.

# 4.0 Research Approach

The research approach is detailed below in eight project tasks:

### Task 1. Develop Curriculum and Basic Theoretical Materials

A project curriculum, including 12 courses was developed, listed below.

1. Introduction to Mine Ventilation
2. Automation
3. Bulk Air Heating and Cooling
4. Diesel Particulate Matter
5. Dust
6. Fans
7. Gaseous Contaminants
8. Modeling and Simulation
9. Underground Coal Ventilation Design
10. Underground Metal/Non-metal Ventilation Design
11. Ventilation Economics
12. Ventilation Surveys

The broad subject matter was developed during the research proposal process and is based on several experiences:

1. Discussion at 15 years of Underground Ventilation Committee Meetings (SME)
   These one to two hour meetings are held every year at the annual SME meetings, and every other year at the North American Mine Ventilation Symposia. They are well attended and generally include rigorous discussion as to the state of the technical community.
2. Kray Luxbacher's experience in teaching a junior level ventilation course to undergraduates (since 2007), and discussing experience in industry with alumni (e.g., what they were glad they knew, what they felt was lacking).
3. Dan Stinnette's years of experience as an international consultant specializing in mine ventilation. This includes deficiencies he saw in the practice, as well as what ventilation engineers told him were strengths and weaknesses in general technical knowledge in the mine ventilation community.

With the exception of *Introduction to Ventilation* (which should be reviewed first) the courses are designed to stand alone and do not have to be worked through in a particular sequence. In areas of overlap, the user is referred to another course or material is restated. For instance, stain tubes are discussed in *Diesel Particulate Matter* and *Gaseous Contaminants*, while in *Ventilation Surveys* users are referred to psychrometry in *Introduction to Ventilation* when the calculation of psychrometric properties of air are discussed (they are also provided with a calculation spreadsheet). Additionally, the courses are asynchronous.

## Task 2: Identify and visit operations around the world practicing the state-of-the-art in mine ventilation.

An index and schedule of visits conducted under the project is provided in Appendix A.

## Task 3. Organize all information gathered and translate into the US context.

All the information that was gathered was organized into a US context, and standard American terminology (e.g., reference to *gob* instead of *goaf,* etc.)

## Task 4. Format the materials for open sourced web based learning

General organization for each module is shown below in Figure 1. Users are expected to take different paths through the courses. Users can always come back to courses they have completed, and they can explore a particular topic further through external open sources, such as NIOSH papers, MSHA resources, and Malcolm McPherson's free online ventilation book. Topic specific external links are provided. Finally, users can assess their learning through the online assessment for every course.

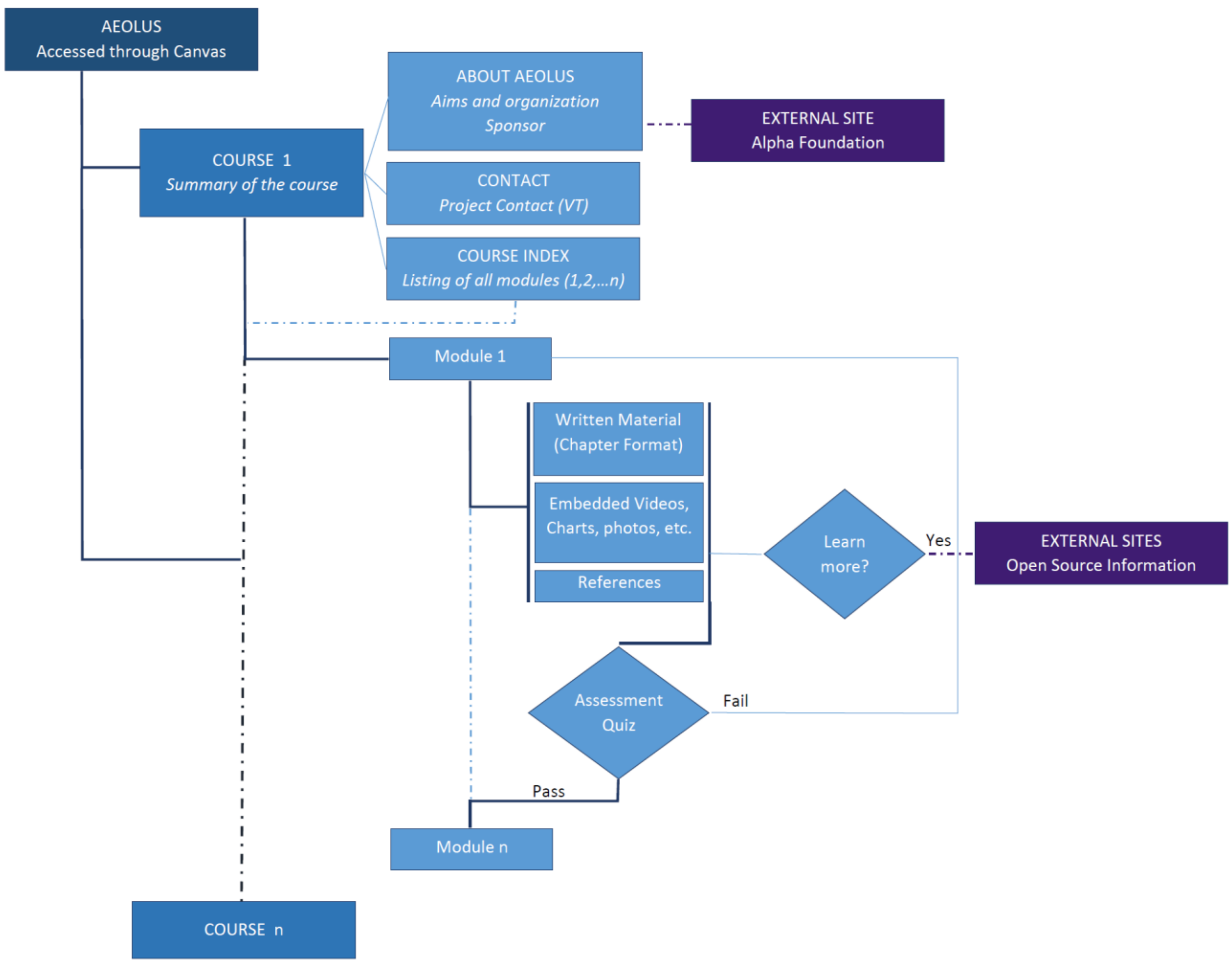



**Figure 1.** General organization and flow for each course in the platform.

Select course content is available in Appendix C, and gives a sense of how a user may move through a course.

## Task 5. Deploy the curriculum on a limited basis for expert assessment

The completed curriculum has been vetted via at least one expert assessment for each course. However, we would prefer to have two and will continue to request these beyond the project end date.

## Task 6. Refine material and debug LMS based on expert feedback in previous task

Minor work during this task included repairing broken links and addressing formatting inconsistencies, or poor feedback on test questions. Further issues from expert reviewers were addressed as detailed below.

1. Intro to Mine Ventilation
   - Added a brief history
2. Automation
   - No major changes, plans to request more expert review
3. Bulk Air Heating and Cooling
   - Add more quiz questions to give more comprehensive coverage of material.
4. Diesel Particulate Matter
   - No major changes, plans to request more expert review
5. Dust
   - Add material under “Introduction”.
   - Consider adding more visual media
6. Fans
   - Add more quiz questions
7. Gaseous Contaminants
   - Add a description of working levels and TLVs
8. Modeling and Simulation
   - Add detail on how to incorporate fan measurements
9. Underground Coal Mine Ventilation
   - Requires further review
10. Underground Metal/Non-metal Ventilation
    - Might add a link to Vitukuri's curve or paper for dust / air velocities.
    - Make mention of battery powered equipment as an option
11. Ventilation Economics
    - Needs more review
    - Style not in line with other courses
12. Ventilation Surveys
    - No major changes

Overall, the feedback thus far is positive. One user indicated he would like to see more links to external resources and other courses in the platform embedded. We agree and will continue to address this. One user noted that it was difficult to gain access to the courses. We have since developed the landing website, shown in Appendix B and provided in sections above, and this seems to have made access easier.

## Task 7. Deploy the curriculum for full use and publicize the tool.

The curriculum is now deployed for public use. The platform has been publicized via two papers detailing its development at the 2017 SME Annual Meeting in February, and at the 2017 North American Mine Ventilation Symposium (NAMVS) held this June in Golden, CO. Additionally, at the NAMVS, Dan Stinnette and Kray Luxbacher reviewed content one-on-one with several experts and received constructive criticism and overall positive comments. The curriculum was also discussed at the meeting of the Underground Ventilation Committee of the SME held at the symposium (as well as the UVC meeting held in February in Denver, CO) and the UVC has agreed to adopt the platform for maintenance, as we have always intended this to be a dynamic system. We will issue invitations to the UVC Community via SME. Finally, we also intend to send invitations to all US ventilation professors to share with students as classes begin in late August.

A plan for maintenance has been developed with general agreement from the Underground Ventilation Committee (a unit committee the SME) discussed Aeolus again at the North American Mine Ventilation Symposium at their meeting on Monday, June 19, 2017 in Golden, Colorado. The committee is willing to support the platform. We have suggested instituting a committee for that purpose. Members for the first term will be Kray Luxbacher (VT), Dan Stinnette (VT, consultant), and Charles Kocsis (UNR). Responsibilities of this committee will be vetting new information proposed for inclusion, incorporating new information that passes review, developing new test questions as necessary and addressing suggestions for improvement.

We presented a formal proposal for inclusion of this committee in the bylaws of the UVC at the 2018 SME meeting in Minneapolis, MN in February 2018. We feel that during the first three years there may be a relatively higher number of suggestions and changes to AEOLUS, so Luxbacher, Stinnette, and Kocsis will comprise the committee until 2020. Members of the board of the UVC serve a 6-year term, so we will request that one board member chair the AEOLUS committee beginning in 2020 for a 6-year term, with interested members of the UVC serving on the committee. Either Kray Luxbacher or Dan Stinnette (or both) intend to also serve the first term, beginning 2020. There was general agreement to the proposal but the bylaws have not been updated yet.

Charles Kocsis, Assistant Professor of Mining Engineering at University of Nevada, Reno, was asked to join us because of his extensive knowledge of metal mining in the United States and worldwide, as well as his active work with the UVC, and he graciously agreed. Additionally, he will serve as a member of Dan Stinnette's doctoral committee. Mr. Stinnette's doctoral work will examine the development and assessment of free, open source platforms for professional development of engineers and will utilize this work as a case study. He will continue to assess use of the platform beyond the scope of this grant, and anticipates completion in May 2019.

### Task 8: Preliminary assessment and validation

We have responded to expert assessments and decided to elicit a few more. We believe this further improves and helps publicize the platform. In addition to requested expert review, any user can provide reviews at the landing site via an easily accessible google form.

# 5.0 Summary of Accomplishments

Over the past two years, we visited 27 sites in seven countries. During those visits we not only gathered materials and documented leading practices, but we also asked ventilation professionals to give us focused responses as to what is required to improve technical competence in the US (and international) ventilation community.

With these materials and feedback we developed 12 asynchronous courses in mine ventilation, as well as a method for feedback and submission of new material.

Finally, we have developed a plan for maintenance of the platform that ensures the courses remain relevant and that the mine ventilation community engages with them.

# 6.0 Dissemination Efforts and Highlights

We have published two papers (Stinnette et al., 2017a; Stinnette et al. 2017b), which are included in Appendix D. Also, we have disseminated information in person to the SME Ventilation Community, delegates at the 2017 North American Mine Ventilation Symposium, and the SME electronic ventilation community. We intend to continue these dissemination efforts, targeting ventilation professors and attendees at upcoming conferences.

# 7.0 Conclusions and Impact Assessment

In conclusion, we have developed a platform for shared open dissemination of learning materials among a highly specialized technical community, and just as importantly, a plan to keep the platform maintained. There are broad and specific impacts expected, and they are primarily long term.

First, we aim to use the self-guided assessments and registration information over the next year to understanding who is using the platform, and how. We will also continue to incorporate peer review and feedback. With the bulk of the research period dedicated to information gathering and development, more detailed assessment will be the primary thesis in Dan Stinnette's dissertation from the standpoint of engineering and technical education.

Second, we suspect there are broader impacts. Open source tools for mining engineers are fairly scarce, and we feel a similar model could be applied to other technical communities in mining engineering, such as blasting.

# 8.0 Recommendations for Future Work

This project is unique in that we plan to continue work on it throughout our careers. Following Dr. Pierre Mousset-Jones' fine example, the investigators see this project as a career long professional service opportunity and are planning for the following short term future work:

Table 1. Short Term Future Work

| Task | Timeline |
|---|---|
| Request more peer review and incorporate suggestions – up to at least 4/course | 2017-18 |
| Develop guidelines to for the Aeolus maintenance committee for presentation to the SME UVC | February 2018 |
| Designate individual responsibilities for committee members | February 2019 |
| Develop a formal peer review process | February 2018 |
| Develop a formal annual review of the courses | February 2019 |
| Develop a complete set of style guidelines | February 2018 |
| Assess users by location, job, frequency | February 2018 |
| Assess the self-paced assessments – how are users scoring (e.g., too difficult, too easy) | February 2018 |
| Consider developing a method for demonstrating a degree of completion of all courses. | February 2019 |

# 9.0 References


Canvas Learning Management System. (2017). Instructure. https://www.canvaslms.com/try-canvas?lead_source_description=Search_Paid,Google&keyword=instructure%2520by%2520canvas&network=g&creative=161701135126&matchtype=e&device=c&model=&creative=161701135126&placement=&position=1t1&gclid=CjsKDwjw5arMBRDz9cK2uen9ORIkAAqmJexPG61HUMofXV8ZUeYsjZ1Vxc_VOOsgjyqMwYsc_TMjGgK2NvD_BwE

Stinnette, J.D., Jong, E.C., Luxbacher, K.D., and Schafrik, S.J. (2017a.) An innovative online platform for the education of mine ventilation professionals. SME Annual Meeting Preprints. February 19-22, 2017; Denver, CO.

Stinnette, J.D., Jong, E.C., Luxbacher, K.D., and Schafrik, S.J. (2017b.) Building a global paradigm for ventilation best-practices. The 16th North American Mine Ventilation Symposium Proceedings. Ed., Brune, J., Golden, CO, Colorado School of Mines.

# 10.0 Appendices

**Appendix A**

# Index and Schedule of Site Visits

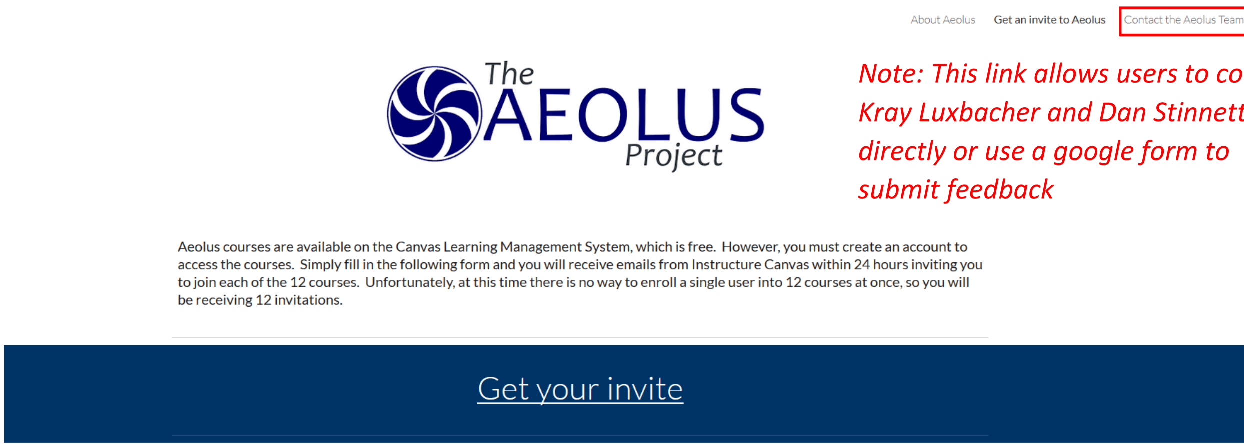
| Date | Visit | Location | Researchers |
|---|---|---|---|
| 3/8/2016 | Quant | Los Andes, Chile | Luxbacher, Stinnette |
| 3/9/2016 | Codelco Andina Mine | Los Andes, Chile | Luxbacher, Stinnette |
| 3/11/2016 | Codelco Chuquicamata Mine | Calama, Chile | Luxbacher, Stinnette |
| 3/14/2016 | Codelco El Teniente Mine | Rancagua, Chile | Luxbacher, Stinnette |
| 3/15/2016 | Universidad de Santiago | Santiago, Chile | Luxbacher, Stinnette |
| 5/17/2016 | Anglo Gold Ashanti Mponeng Mine | Northwest Provence, South Africa | Schafrik, Stinnette |
| 5/18/2016 | Bluhm Burton Engineering | Johannesburg, South Africa | Schafrik, Stinnette |
| 5/19/2016 | Impala Platinum Mine | Rustenburg, South Africa | Schafrik, Stinnette |
| 5/20/2016 | University of Witswaterstrand | Johannesburg, South Africa | Schafrik, Stinnette |
| 5/23/2016 | University of Pretoria | Pretoria, South Africa | Schafrik, Stinnette |
| 5/31/2016 | Cetamin Training Facility | Lima, Peru | Luxbacher |
| 5/31/2016 | ISEM Offices | Lima, Peru | Luxbacher |
| 6/3/2016 | Cerro Lindo Mine | Chavin, Chincha, Peru | Luxbacher |
| 11/3/2016 | University of New South Wales | Sydney, Australia | Schafrik, Stinnette |
| 11/4/2016 | Peabody Metropolitan Colliery | Australia | Schafrik, Stinnette |
| 11/7/2016 | Peabody Wambo Colliery | Singleton, Australia | Schafrik, Stinnette |
| 11/8/2016 | Curtin University | Perth, Australia | Schafrik, Stinnette |
| 11/8/2016 | AusIMM (Perth) | Perth, Australia | Schafrik, Stinnette |
| 11/11/2016 | BHP Billiton Olympic Dam Mine | South Australia | Schafrik, Stinnette |
| 11/14/2016 | Chasm Consulting | Perth, Australia | Schafrik, Stinnette |
| 11/16/2016 | Rick Brake | Perth, Australia | Schafrik, Stinnette |
| 4/19/2017 | Atlas Copco | Orebro, Sweden | Luxbacher, Stinnette |
| 4/23/2017 | Zinkgruvan Mine | Zinkgruvan, Sweden | Luxbacher, Stinnette |
| 4/24/2017 | ABB | Vasteras, Sweden | Luxbacher, Stinnette |
| 4/22/2017 | CFT | Gladbeck, Germany | Luxbacher, Stinnette |
| 4/24/2017 | TLT Turbo | Zweibrucken, Germany | Luxbacher, Stinnette |
| 6/30/2017 | Freeport-McMoRan Henderson Mine | Colorado, USA | Stinnette |

**Appendix B**

# Accessing and Navigating Aeolus Courses

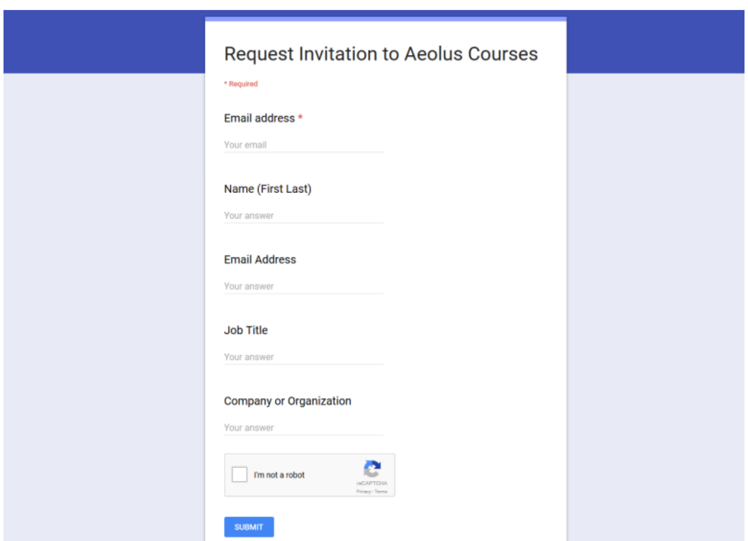
User visits the Aeolus landing website:

https://sites.google.com/view/aeoluslaunch/get-an-invite-to-aeolus

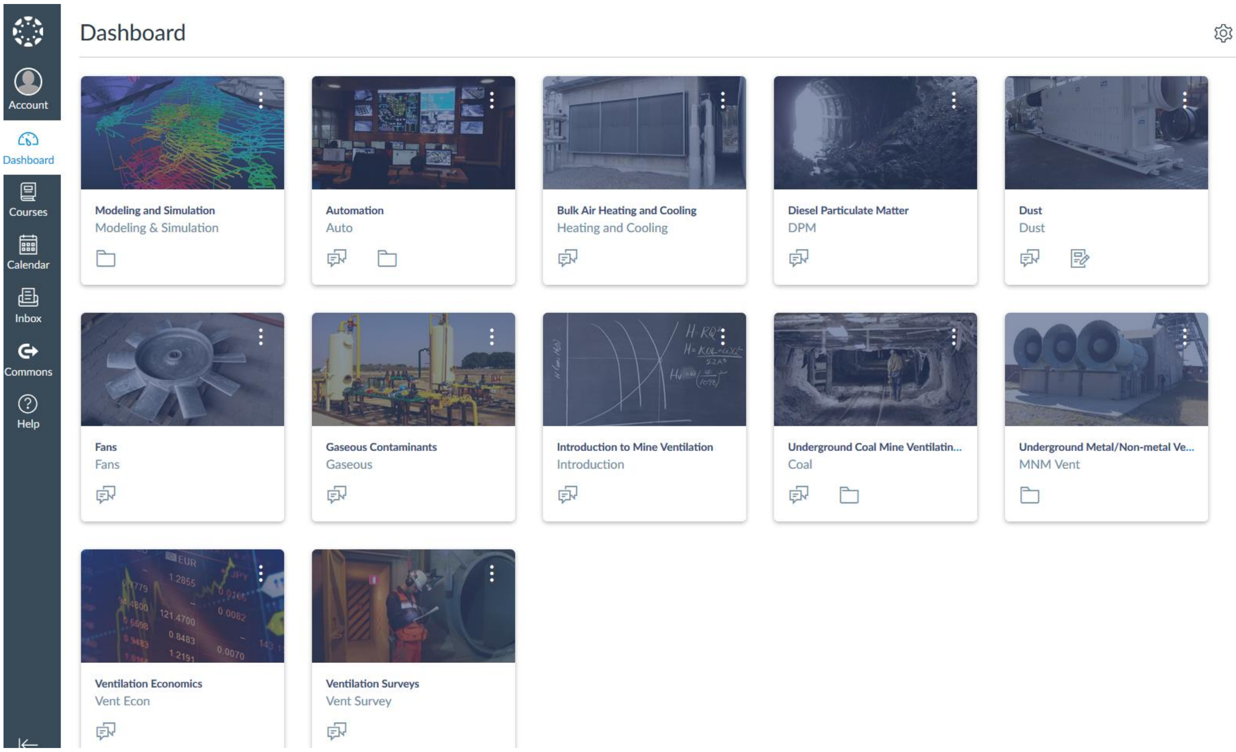



*Note: This link allows users to contact Kray Luxbacher and Dan Stinnette directly or use a google form to submit feedback*

User clicks “Get your invite”:




Canvas Instructure invitations will arrive to the user via email. Once they click accept and enroll in Canvas, which requires name, email, and password, they will see the following course landing page when they log in:




The user clicks a tile to get started with a particular course, and comes to a landing page that is similar for each course:


From this page, the user can navigate the course sequentially ("Get Started") or go to a specific topic ("Go to the Course Index").

**Appendix C**

# Course Content

Pages in order for the 12 courses are included:

- Introduction to Mine Ventilation
- Automation
- Bulk Air Heating and Cooling
- Diesel Particulate Matter
- Dust
- Fans
- Gaseous Contaminants
- Modeling and Simulation
- Underground Coal Ventilation Design
- Underground Metal/Non-metal Ventilation Design
- Ventilation Economics
- Ventilation Surveys

These courses along with the others are optimized for viewing online. Get an invitation to the courses at: https://sites.google.com/view/aeoluslaunch/get-an-invite-to-aeolus

# Introduction to Mine Ventilation

Jump to Today Edit

## Introduction to Mine Ventilation

This module will introduce the user to mine ventilation including a brief history, supporting theory, and common terms.  This module concentrates on incompressible flow and introduces measurements and calculation for psychrometry, describes fluid flow in mine ventilation system and the measurement and estimation of friction and shock losses for head loss calculations.  Additionally, the module describes the Square Law (Atkinson Equation), and the conditions under which it holds.

**Learning Objectives**

1. **Articulate the historical development of mine ventilation as a technical field.**
2. **Calculate psychrometric properties of air and appreciate their significance.**
3. **Describe the theory that underpins the practice of mine ventilation as related to incompressible flow.**
4. **Define terms commonly used in the US and global mine ventilation contexts.**

Get Started

Go to the Course Index

About Aeolus

# Course Summary:

| Date | Details |
| --- | --- |
| | history (https://canvas.instructure.com/courses/1068093/assignments/5225671) |

# About Aeolus

The Aeolus Project is a suite of open access mine ventilation courses developed for the purposes of improving competence and encouraging a dynamic professional mine ventilation community. These courses are designed to provide users with a single repository for modern mine ventilation knowledge compiled from a variety of international sources, including, but not limited to, published academic references, mine ventilation specialists, and ventilation equipment manufacturers. While the intended audience is US mining engineers we aim to provide leading international practice here, and to engage a global audience. The Aeolus Project is intended to be a living, open access platform. If you have suggestions or questions please contact us at: aeolusmining@gmail.com.

The information presented in the mine ventilation curriculum was developed by a grant awarded from the **Alpha Foundation for the Improvement of Mine Safety and Health, Inc.** **(http://www.alpha-foundation.org/)** (Alpha Foundation) to **Virginia Tech** **(http://www.vt.edu)**. The content was developed by the authors and does not reflect the official policies of the Alpha Foundation. Additionally, the mention of trade names, commercial practices, or organizations does not imply endorsement by the Alpha Foundation or its directors and staff.

*Note: The "About Aeolus" page is included with every course, but is the same with every course, so it is only included in this print Appendix once.*

# 1.0. Introduction to Mine Ventilation Engineering

**Why do we ventilate mines? In short, to ensure health and safety by controlling the following:**

- Mine Gases
- Dusts
- Fumes
- Heat/Cold/Humidity
- Oxygen

In ventilation engineering our primary concerns are air quality and air quantity. The following control processes are implemented:

1.Prevention or Avoidance
2.Removal or Elimination
3.Suppression or Absorption
4.Containment or Isolation
5.Dilution or Reduction

(**Hartman et al.,1997 (https://canvas.instructure.com/courses/1068093/pages/Resources?titleize=0)** )

In theory, designing a mine ventilation system is much like designing an HVAC system except the materials we use to construct our duct are heterogeneous, and difficult to characterize.

Figure 1. An HVAC ventilation system versus a mine ventilation system. The mine system is obviously more difficult to characterize and control.

# 2.0. A Brief History of Mine Ventilation

A brief history of mine ventilation is instructive. First, the field is quite old, and the development of technology has been closely linked to changes in society (industrialization) and labor practices. Major accidents have also driven technology, as well as US ventilation regulation.

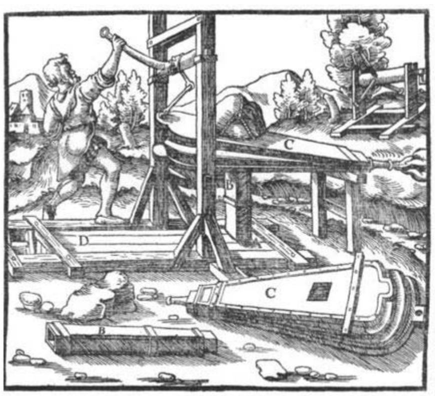
4000 B.C.
European miners develop chalk tunnels looking for flint. Evidence they built fires in working faces to weaken rock, likely that they also noticed effects on ventilation.

1200 B.C.

600 B.C.
The Laurium Silver Mines in Greece show evidence of divided shafts and multiple airways feeding sections. Also, slaves used palm fronds to fan air.

1586 A.D.
Georgius Agricola publishes De Re Metallica.

1700-1820 A.D.
Attempts are made to design light sources that will not cause explosions



A—Smaller part of shaft. B—Square conduit. C—Bellows. D—Larger part of shaft.

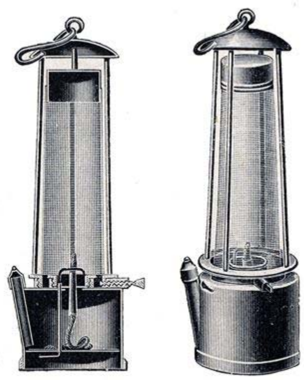


Fig. 192. Davysche Sicherheitslampe


1850 A.D.
Power driven ventilation is implemented

Working conditions are legislated.
1900 A.D.

1930 A.D.
Implementation of the axial mine fan

Analog computer assists in vent planning
1952 A.D.

1969 A.D.
Consol No. 9 Mine Fire
Federal Coal Mine Health & Safety Act is signed (amended '77)

Sago and Darby Explosions, Aracoma Fire, MINER Act is signed
2006 A.D.

Upper Big Branch Explosion
2010 A.D.

# 3.0. Psychrometry

**Psychrometry defined**

(sī′käm·ə·trē) The science and techniques associated with measurements of the water vapor content of air or other gases.

Psychrometry is critical to understanding and designing ventilation system operation because, after all, it is air that we seek to measure and control, so we must understand its properties. Psychrometry is utilized in understanding:

- Mechanical movement of air
- Air cooling and heating
- Strata heat
- Effects on the human body
- Effects of humidity on strata
- Hygroscopic strata

The composition of dry air is given as follows (McPherson, 2009):

Table 1. Composition of Dry Air

| Gas | Percent by Volume | Percent by Mass | Molecular weight (g/mol) |
|---|---|---|---|
| Nitrogen | 78.03 | 75.46 | 28.015 |
| Oxygen | 20.99 | 23.19 | 32.000 |
| Carbon Dioxide* | 0.038 | 0.05 | 44.003 |
| Hydrogen | 0.01 | 0.0007 | 2.016 |
| Monatomic Gases | 0.94 | 1.30 | 39.943 |
| Dry Air | | | 29.966 |

**Atmospheric $CO_2$ may vary significantly.*

To calculate the properties of air the ventilation engineer must establish a state point measurement, which includes the wet bulb temperature (Tw), the dry bulb temperature (Td), and the barometric pressure (Pb).

*Definition of heat terms*

***Latent Heat***
*Rise in heat content that occurs due to evaporation increasing energy content of the air/vapor mixture. The rise in heat content does not result in a rise in temperature.*

***Sensible Heat***
*Rise in heat content that does result in a rise in temperature*

Dry Air and Water Vapor

Under natural conditions, air contains water vapor. Thermodynamically, consider some fraction of dry air plus some fraction of water vapor behaving as a gas, such that;

Pt = Pa + Pv

So, according to Dalton's Law, the total pressure (Pt) exerted is the sum of the partial pressures (ideal gases).

# 3.1. Psychrometric Calculations and Definitions

*Note: all equations are given in SI (left) and English (right) units. For an excellent derivation and discussion of these calculations (SI only) see* [McPherson (2009) (https://canvas.instructure.com/courses/1068093/pages/resources)](https://canvas.instructure.com/courses/1068093/pages/resources) .

**Saturation Vapor Pressure**

If the pressure of water vapor is increased in a given space the vapor will eventually condense. The pressure, es, exerted by water at saturation conditions is:

$$e_{sw} = 610.6\ exp\left[\frac{17.27t_w}{237.3 + t_w}\right] \qquad e_{sw} = 0.18079\ exp\left[\frac{17.27t_w - 552.64}{395.14 + t_w}\right]$$

*(Pa)* *(In Hg)*

Use the wet bulb temperature to calculate saturation vapor pressure. Use the dry bulb temperature to calculate saturation vapor pressure for input (esd) into the relative humidity equation.

**Saturated Moisture Content**

Gives the moisture content at saturation.

$$X_s = 0.622\frac{e_{sw}}{(P - e_{sw})} \qquad X_s = 0.622\frac{e_{sw}}{(P - e_{sw})}$$

*(kg/kg dry air)* *(lb/lb dry air)*

**Moisture Content (Specific Humidity)**

Gives the mass of water per mass of dry air.

$$X = \frac{S - 1005t_d}{L_w + 1884(t_d - t_w)} \qquad X = \frac{S - 0.24(t_d - 32)}{L_w + 0.45(t_d - t_w)}$$

$$X = \frac{L_w X_s - 1005(t_d - t_w)}{L_w + 1884(t_d - t_w)} \qquad X = \frac{L_w X_s - 0.24(t_d - t_w)}{0.45(t_d - t_w) + L_w}$$

*(kg/kg dry air)* *(lb/lb dry air)*

**Latent Heat of Evaporation**

Latent heat of evaporation is the heat required to evaporate 1 kg (or 1 lb) or water.

$$L_w = (2502.5 - 2.386t_w)1000 \qquad L_w = (1094.1 - 0.5699t_w)$$

*(J/kg)* *(Btu/lb)*

**Actual Vapor Pressure**

The actual pressure of the vapor in the air (not necessarily saturated)

$$e = \frac{PX}{0.622 + X} \qquad e = \frac{PX}{0.622 + X}$$

*(Pa)* *(In Hg)*

**Density**

Apparent is on a dry air basis, while actual is based on the actual air saturation.

$$\rho_m\,(apparent) = \frac{(P-e)}{287.04(t_d+273.15)} \qquad \rho_m\,(apparent) = \frac{1.325(P-e)}{(t_d+459.67)}$$

*(kg dry air/m³)* *(lb dry air/ft³)*

$$\rho_m\,(actual) = \frac{(P-0.378e)}{287.04(t_d+273.15)} \qquad \rho_m\,(actual) = \frac{1.325(P-0.378e)}{(t_d+459.67)}$$

*(kg moist air/m³)* *(lb moist air/ft³)*

The apparent density of air decreases as moisture content rises.

**Sigma heat** is a term that is generally, unique to mining.

Sigma heat is similar to enthalpy, but ignores sensible heat of liquid water (sensible heat lost by air is balanced by latent heat gained by air – only dependent on tw).

Allows mining engineers to evaluate thermal additions and losses from an airstream.

$$S = L_w X_s + 1005t_w \qquad S = L_w X_s + 0.24(t_w - 32)$$

*(J/kg dry air)* *(Btu/lb dry air)*

**Enthalpy**

Total heat content (sensible and latent)

$$H = S + (4187t_w X) \qquad H = S + ((t_w - 32)X)$$

*(J/kg dry air)* *(Btu/lb dry air)*

**Relative Humidity**

Usually referenced as a measure of comfort – the degree of saturation as the ratio of actual vapor pressure to saturation vapor pressure (%).

$$rh = \frac{e}{e_{sd}} \times 100$$

See an example calculation...

# 3.2. Psychrometric Calculation Example

At a point in a mine a wet bulb temperature of 50°F and a dry bulb temperature of 67°F are measured. The barometric pressure is 27.33 in Hg. What is the relative humidity?

1. Calculate the saturation vapor pressure (wet basis):

$$e_{sw} = 0.18079\, exp\left[\frac{17.27t_w - 552.64}{395.14 + t_w}\right] = 0.18079\, exp\left[\frac{17.27(50) - 552.64}{395.14 + (50)}\right] = 0.3635\ in.Hg$$

2. Calculate the moisture content (at saturation):

$$X_s = 0.622\frac{e_{sw}}{(P - e_{sw})} = 0.622\frac{0.3635}{(27.33 - 0.3635)} = 0.00838\frac{lb}{lb}\,dry\ air$$

3. Calculate the latent heat of evaporation:

$$L_w = (1094.1 - 0.5699(t_w)) = (1094.1 - 0.5699(50)) = 1065.6\ Btu/lb$$

4. Calculate the actual moisture content:

$$X = \frac{L_w X_s - 0.24(t_d - t_w)}{0.45(t_d - t_w) + L_w} = \frac{(1065.6)(0.00838) - 0.24(67 - 50)}{0.45(67 - 50) + 1065.6} = \frac{0.00452 lb}{lb}\,dry\ air$$

5. Calculate the vapor pressure:

$$e = \frac{PX}{0.622 + X} = \frac{(27.33)(0.00452)}{0.622 + 0.00452} = 0.1973\ in.Hg$$

6. Calculate the saturation vapor pressure (dry basis):

$$e_{sd} = 0.18079\, exp\left[\frac{17.27(67) - 552.64}{395.14 + 67}\right] = 0.6687\ in.Hg$$

7. Calculate the relative humidity:

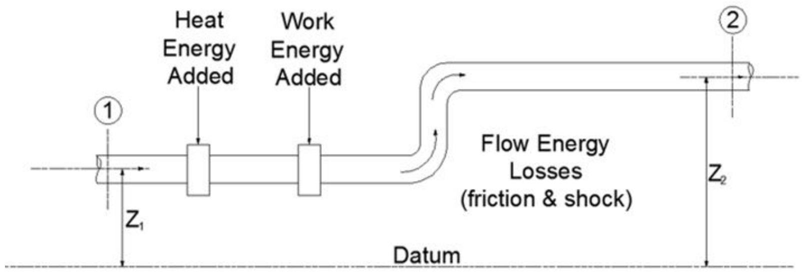
$$rh = \frac{e}{e_{sd}} \times 100 = \frac{0.1973}{0.6687} \times 100 = \boxed{29.5\%}$$

An excel sheet can easily be created for input of a state point in SI or English units and calculation of the psychrometric equations, as shown below.

| | | | |
|---|---|---|---|
| INPUT | | | |
| wet bulb temperature | tw | 50 | °F |
| dry bulb temperature | td | 67 | °F |
| barometric pressure | P | 27.33 | Pa |
| | | | |
| CALCULATIONS | | | |
| Saturation vapor pressure (wet) | esw | 0.36346 | in. Hg |
| Saturation vapor pressure (dry) | esd | 0.66866 | in. Hg |
| Moisture Content (Specific Humidy) | Xs | 0.00838 | lb/lb dry air |
| Latent Heat of Evaporation | Lw | 1065.60500 | Btu/lb |
| Sigma Heat | S | 13.25351 | Btu/lb dry air |
| Moisture Content | X | 0.00452 | lb/lb dry air |
| Moisture Content | X | 0.00452 | lb/lb dry air |
| Actual vapor pressure | e | 0.19727 | in. Hg |
| Apparent Density | ρm (app) | 0.06875 | lb dry air/ft3 |
| Actual Density | ρm (act) | 0.06857 | lb moist air/ft3 |
| Enthalpy | H | 13.33491 | Btu/lb dry air |
| Relative Humidity | rh | 29.50201 | % |

# 4.0. Flow in Mines

Airflow through mine openings and ducts is based on the general energy equation, shown in the simple pipe example below:




$$(Total\ Energy)_1 = (Total\ Energy)_2 + (Flow\ Energy\ Losses)_{1\text{-}2}$$

and Bernoulli's equation as applied to incompressible fluids, where:

$$\frac{p_1}{w} + \frac{V_1^2}{2g} + Z_1 = \frac{p_2}{w} + \frac{V_2^2}{2g} + Z_2 + H_l$$

$$\frac{p}{w} = \ Static\ Energy \qquad \frac{V^2}{2g} = \ Velocity\ Energy$$

$$Z = Potential\ Energy \qquad H_l = \ Flow\ Energy\ Loss$$

and, finally, the general energy equation, where:

$$\underbrace{H_{s_1} + H_{v_1} + H_{z_1}}_{H_{t_1}} = \underbrace{H_{s_2} + H_{v_2} + H_{z_2}}_{H_{t_2}} + H_l$$

*A note on head versus pressure:*

**Pressure** is force exerted per unit area
Common Units: psi or Pa

**Head** is height of a column of water or mercury equivalent to pressure exerted by air
Common Units: in or mm

p = wH

Where, H = head (in. water or mm water [or Hg])

p = pressure (psi or Pa)

w = specific weight (lb/ft$^3$ or kg/m$^3$)

Harman and others express these differences best:

*“There are inconsistencies in practice…in barometric pressure measurements, the equivalent height of a column of mercury (in in. or mm) is usually*

*used…in expressing head, inches of water is used as the English unit, but Pa as the SI unit..” (**[Hartman et al., 1997 (https://canvas.instructure.com/courses/1068093/pages/resources)](https://canvas.instructure.com/courses/1068093/pages/resources)** ).*

*Modified Energy Equation*

$$H_{s_1} + H_{v_1} = H_{s_2} + H_{v_2} + H_l$$

$$H_{t_1} = H_{t_2} + H_l$$

If a gage pressure is employed for all static pressures, potential energy (*Hz*) can be omitted from the above equation, and we are left with:

$$H_l = H_f + H_x$$

$$H_f = Friction\ loss$$

$$H_x = Shock\ loss$$

*Friction losses* are due to flow through ducts of constant area, while *shock losses* occur when flow changes direction or the area of the duct changes.

So now we must understand how to determine friction and shock losses, which at best can be measured directly for sections of a mine and applied carefully.  If you have not been able to measure friction loss directly you may begin with losses provided in the literature.  Remember, with mine openings as ducts we have a much less uniform material to work with than the average HVAC engineer.

Figure 4.1. Typical mine openings (right), compared with a more uniform HVAC duct.  It is evident that the mine openings have relatively high frictional losses.

**Characterizing head losses**

***It is important that a ventilation engineer can measure and estimate head losses, including friction, shock, and velocity.***

Direct measurement of losses are described in the Surveying course.  Values are also provided in the literature for preliminary work.

View values determined by McPherson (2009) in English and SI units: **Friction_Factors.pdf (https://canvas.instructure.com/courses/1068093/files/53283577/download?wrap=1)** **(https://canvas.instructure.com/courses/1068093/files/53283577/download?wrap=1)**

**Minimize File Preview**

ZOOM

| | Friction factor k [(lb·min²/ft⁴) x10$^{-10}$] | Friction factor k (kg/m³) | Coefficient of f f (dimensionl |
|---|---|---|---|
| **Rectangular Airways** | | | |
| Smooth concrete lined | 22 | 0.004 | 0.0067 |
| Shotcrete | 30 | 0.0055 | 0.0092 |
| Unlined with minor irregularities only | 49 | 0.009 | 0.015 |
| Girders on masonry or concrete walls | 51 | 0.0095 | 0.0158 |
| Unlined, typical conditions no major irregularities | 65 | 0.012 | 0.02 |
| Unlined, irregular sides | 75 | 0.014 | 0.023 |
| Unlined, rough or irregular conditions | 86 | 0.016 | 0.027 |
| Girders on side props | 102 | 0.019 | 0.032 |
| Drift with rough sides, stepped floor, handrails | 216 | 0.04 | 0.067 |
| **Steel Arched Airways** | | | |
| Smooth concrete all round | 22 | 0.004 | 0.0067 |

Shock losses can be due to turns in the system, changes in cross sectional area, and ventilation controls, such as overcasts. Shock losses can also be measured directly (especially for lines of overcasts), calculated via McPherson's method (2009) - see Appendix, Ch. 5. (https://canvas.instructure.com/courses/1068093/files/53283593/download?wrap=1) (https://canvas.instructure.com/courses/1068093/files/53283593/download?wrap=1) (https://canvas.instructure.com/courses/1068093/files/53283593/download?wrap=1), or estimated via the method of Hartman et al. (1997) (https://canvas.instructure.com/courses/1068093/pages/resources), shown in the Table below:

Table 4.1. Estimation of shock loss by equivalent length (after Hartman et al., 1997 (https://canvas.instructure.com/courses/1068093/pages/resources))

| Source | Le (ft) | Le (m) | Source | Le (ft) | Le (m) |
|---|---|---|---|---|---|
| Bend, acute, round | 3 | 1 | Contraction, gradual | 1 | 1 |
| Bend, acute, sharp | 150 | 45 | Contraction, abrupt | 10 | 3 |
| Bend, right, round | 1 | 1 | Expansion, gradual | 1 | 1 |
| Bend, right, sharp | 70 | 20 | Expansion, abrupt | 20 | 6 |
| Bend, obtuse, round | 1 | 1 | Splitting, straight branch | 30 | 10 |
| Bend, obtuse, sharp | 15 | 5 | Splitting, deflected branch (90°) | 200 | 60 |
| Doorway | 70 | 20 | Junction, straight branch | 60 | 20 |
| Overcast | 65 | 20 | Junction, deflected branch (90°) | 30 | 10 |
| Inlet | 20 | 6 | Mine car or skip (20% of airway) | 100 | 30 |
| Discharge | 65 | 20 | Mine car or skip (40% of airway) | 500 | 150 |

This table approximates Le based on airways where : K = 100 x $10^{-10}$ lb×min$^2$/ft$^4$ (0.186 kg/m$^3$)

Rh=A/O=2 ft (0.61 m)

w = 0.0750 lb/ft$_3$ (1.201 kg/m$^3$).

Finally, in order to characterize velocity head loss, pressure head is a measure of a fluid column, and has units of length. More generally, it is a measure of the internal energy of fluid exerted on its container. In the following explanation pv has units of psf and v has units of fps. We treat the first equation below such that in the second equation v is in fpm and Hv is in in. H2O:

$$p_v = \frac{\omega v^2}{2g}$$

$$H_v = \frac{\omega v^2}{\left(5.2\,\frac{psf}{in\ H2O}\right) 2\left(32.2\ \frac{ft}{s^2}\right)\left(60\,\frac{s}{min}\right)^2} = \frac{\omega v^2}{1205568} = \omega\left(\frac{v}{1098}\right)^2$$

Further, at standard conditions, where w=0.0750 lb/$ft^3$

$$H_v = \frac{(0.0750\ \frac{lb}{ft^3})v^2}{\left(5.2\,\frac{psf}{in\ H2O}\right) 2\left(32.2\ \frac{ft}{s^2}\right)\left(60\,\frac{s}{min}\right)^2} = \left(\frac{v}{4009}\right)^2$$

# 5.0. The Square Law

When calculating incompressible flow in underground mines we rely on the square law or Atkinson equation:

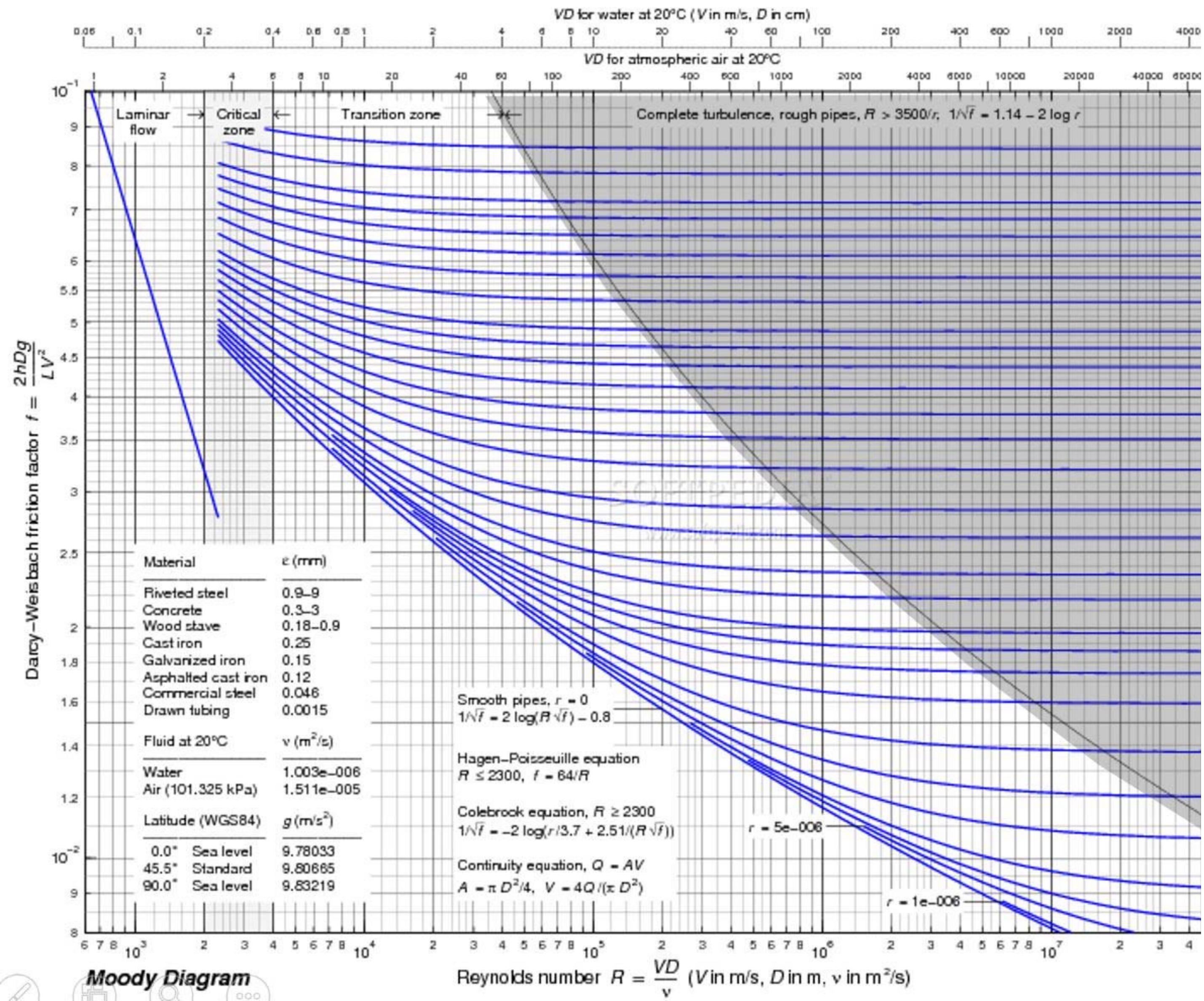
$$H_l = RQ^2$$

where Hl is head loss (friction and shock), R is resistance, and Q is quantity. Resistance is calculated as:

$$R = \frac{KO\,(L + L_e)}{5.2\,A^3}$$

where, K is friction factor, L is length of the airway, Le is the equivalent length shock loss, and A is the cross sectional area of the airway. This calculation is for English units. When working in SI units, remove 5.2 from the equation.

It is important to note that the **square law only applies to fully developed turbulent flow**. In the transitional zone, H = RQ$^n$, and n can vary from 1.8 to 2.5 ([McPherson, 2009 (https://canvas.instructure.com/courses/1068093/pages/resources)](https://canvas.instructure.com/courses/1068093/pages/resources)). The Moody Diagram below designates fully developed turbulent flow in gray.



# Automation

Jump to Today Edit

## Automation

Automation of ventilation systems can result in substantial savings, improved health and safety outcomes, as well as faster emergency response. This module covers different levels of automation as well as practical advice on implementation of automated ventilation systems and health and safety issues related to ventilation automation. These concepts are primarily applied to metal and nonmetal mines in the U.S and are often referred to as ventilation-on-demand (VOD).

**Learning Objectives**

1. **Describe the levels of automated ventilation systems that may be implemented in a mine.**
2. **List health and safety gains and concerns that may be associated with VOD.**
3. **Determine appropriate sensing and layout for a VOD system.**
4. **Describe the other systems that may interact significantly with VOD (e.g., tracking).**

Get Started

About the Aeolus Project

# Course Summary:

| Date | Details |
|---|---|

# 1.0 Introduction to Ventilation Automation

Automation of all mining systems, including the ventilation system, is rapidly emerging and evolving. In the U.S. context, there is relatively little high level automation of ventilation systems in underground coal mines, primarily due to the regulatory atmosphere. **30 CFR §75.324 (https://canvas.instructure.com/courses/1186482/pages/references)** details the requirement that ventilation in underground coal mines shall not be changed more than 9,000 cfm without removal of electrical power and nonessential personnel, which, for practical purposes, excludes use of ventilation-on-demand (VOD) in US underground coal mines. The following course will, thus, focus on the use of VOD for metal and non-metal mining. It is instructive to recall that the primary responsibility of a ventilation engineer is to provide an adequate quantity and quality of air. With the advent of automated mining and haulage systems we can begin to design for "adequate" quantity and quality a bit differently, depending on where mining is active and which areas of the system humans are interacting with.

# 1.1. What is Ventilation-on-Demand?

A ventilation-on-demand (VOD) system is designed to send air only where it is needed, and can result in substantial savings. Demand may be based on:

- Lowering temperature to safe working conditions for miners.
- Raising the temperature to safe working conditions for miners and to prevent ice buildup in key areas (e.g., shafts).
- Lowering concentration of contaminants for miner health (e.g., dust, DPM, $H_2S$).
- Lowering concentration of contaminants to reduce explosions risk (e.g., methane).
- Emergency response scenarios such as fires in key areas or a "fail safe" full ventilation strategy.

Advantages and disadvantages of VOD are detailed below. Due to the high capital cost and considerable on-site expertise needed most complex VOD systems are currently installed in large metal/nonmetal operations. However, as automation becomes more ubiquitous in our everyday lives and in mines, and as "plug and play" components become more economical they will begin to appear in smaller mines.

| **Advantages** | **Disadvantages** |
|---|---|
| Can save large amounts of money/energy | System failure can result in safety/health hazards |
| Allows ventilation system to function more efficiently | Costly to install |
| Allows for remote, continuous collection of atmospheric information for improved maintenance and trouble shooting | Must be updated, calibrated, and maintained continuously |

# 1.2. Levels of VOD

[Acuna and Allen (2017) (https://canvas.instructure.com/courses/1186482/pages/references)](https://canvas.instructure.com/courses/1186482/pages/references) present five levels of VOD, orginally proposed by Tran-Valade and Allen (2013), that may be used as stand alone methods or in combination with each other. They are:

**Level 1: User Control (Manual Control)**

In Level 1 specific control points are designed for sections of the mine (or the entire mine), which may be controlled from the surface. For instance, a "production point" might raise the ventilation to a set quantity for the dilution of strata gas or dust.

**Level 2: Time of Day Scheduling**

Time of Day Scheduling uses set points from Level 1 along with timers to automate the ventilation based on the mine schedule. For instance, in a mine where blasting regularly takes place at the beginning of evening shift, followed by mucking, followed by an idle shift you may have set points similar to those below:

| Time | Event | Air Quantity |
|---|---|---|
| 4:00 pm | Blasting, no personnel underground | low |
| 4:15 pm | Blasting complete, no personnel underground, dilution of fumes/dust | high |
| 4:30 pm | Personnel traveling to work sites | medium |
| 4:45 pm | mucking begins, dilution of dust/DPM | high |

Clearly, this represents a simple version of Time of Day Scheduling, perhaps for a single sections. As one begins to consider multiple sections and mains scheduling becomes more complex.

**Level 3: Event Based**

Event based can include both planned or routine events, such as blasting or emergency scenarios. Emergency scenarios must be carefully planned to account for the safest modes in the event of an emergency. A simple scenario may just ensure that all escapeways have maximum ventilation while others might include the automatic closure of fire doors and integrated tracking of personnel.

These first three levels are fairly simple in that they do not require high level monitoring. One must be able to communicate with and monitor fans, at the very least, but because they are based on specific events or times they can be triggered from a control room or on a timer are are not automatically responsive to the mine atmosphere.

**Level 4: Tagging**

With tagging the level of complexity increases, as now the system will be set to respond to operations in the mine. These operations might be the movement of equipment. For instance, tracking the number of haul trucks on a ramp and increasing air quantity to dilute known DPM per truck. This might also include automatically increasing (or starting) ventilation in areas that miners have entered, based on a personnel tracking system. In any case, at this point the underground monitoring infrastructure becomes more complex and reliability of this infrastructure is critical.

**Level 5: Environmental**

Finally, the most complex level is tracking not only the movement of people and equipment but monitoring and responding to changes in the mine atmosphere, which may include temperature, gas, humidity, dust, and DPM, and may be complicated by the presence of people and equipment. At this point, the underground monitoring infrastructure is complex, the data collected are quite large, and reliability of the system is extremely critical. However, the health and safety benefits, as well as the economic indicators can make such a system advantageous.

# 1.4. Considerations for planning of a VOD system

Included below are important considerations for the planning of a VOD system, particularly a **Level 5 (https://canvas.instructure.com/courses/1186482/pages/1-dot-2-levels-of-vod)** system:

**Do you have the trust of personnel, AND has the system been carefully vetted?**

Anecdotally, the difference between the successful and unsuccessful use of automation systems at mines is buy in from personnel. The premature installation of a system with poor monitoring or performance capability - a system that simply doesn't supply the quality and quantity of air required early in the implementation timeline or one that triggers false alarms - will very quickly frustrate employees, and lower their trust in the system. This is a critical problem and can be address with careful design and implementation, even when it takes longer than expected. Education of the work force is also key. By installing a VOD system it is very likely that less air will enter the mine every day, and first reaction to lowered air requirements may be suspicion. However, if management can detail how the system actually improves quality of air in real time and responds to environmental factors to improve health and safety personnel are more likely to buy in.

**Do you have the appropriate level of technical competence and capacity?**

A complex automated system must be supervised by people (often remotely) even in the best of circumstances. Generally, a person supervising such a system must be well versed in both ventilation and the operation of the entire mine, as well as potential emergency scenarios. They also must have some autonomy in responding to emergency scenarios as they unfold. The level of competence and technical expertise required for such a person is quite high compared to the control room operator of 20 years ago. Additionally, a Level 5 VOD system is producing a "big dataset" at least every day. These data must be presented in a way that is meaningful. For example, if a control room operator acknowledges the same "alert" every minute or so of every day, that datum is no longer a true alert. It is all to easy to exhaust a control room operator, and surpass cognitive capacity, so such systems also must be designed with human factors in mind.

**Can you really justify such a system?**

A ventilation engineer must be able to demonstrate two things with the installation of a Level 5 system:

1) A standard of safety and health that is at least as high as the pre-VOD standard, preferably much better.

2) Savings in energy that offset the capital cost and maintenance costs of such a system.

Generally, this is why these systems are currently limited to large underground metal and nonmetal mines. Certainly, there are a number of case studies justifying VOD, however. The study by Acuna and Allen (2017) which details the five levels of automation cites an expected energy savings at the **Totten**

Mine (http://www.vale.com/en/aboutvale/news/pages/vale-inaugura-primeira-nova-mina-em-sudbury-em-mais-de-40-anos.aspx) of 50% of baseline energy and natural gas consumption, which is impressive. Even more impressive, 25% of the total savings is expected from installation at Level 1 alone.

**Figure 1.1. Arial view of Totten Mine surface facilities, Sudbury, Canada.**

**What are you doing with these data?**

As mentioned previously, data collected under Level 5 systems constitute enormous sets, which must be organized, processed, stored, and backed up. They also must be available in real time to be meaningful.

**How reliable is the system and how do you maintain it?**

As we begin to rely more on automated systems reliability is critical. Data should be closely examined for anomalies or evidence that the system is functioning improperly. All monitoring devices should be regularly examined and tested or calibrated. Finally, because a mine is inherently a dynamic system, a VOD system must also be dynamic, growing and shrinking as the mine develops with timely and best practice installation.

# Bulk Air Heating and Cooling

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**Bulk Air, Heating, and Cooling**

The following course provides an overview of bulk air heating and cooling techniques and strategies. There is a discussion of basic thermodynamic theory, technology and strategic implementation with regard to the heating and cooling of mine air in a variety of applications and scales. This course is part of a larger suite of courses covering a variety of topics in mine ventilation.

These course are designed to provide users with a single repository for modern mine ventilation knowledge compiled from a variety of international sources including but not limited to published academic references, mine ventilation specialists, and ventilation equipment manufacturers.

The information presented in the mine ventilation curriculum was developed by a grant awarded from the Alpha Foundation. The included content was developed by the authors and do not reflect the official policies of the Alpha Foundation. Additionally, the mention of trade names, commercial practices, or organizations does not imply endorsement by the Alpha Foundation.

### Learning Objectives

1. **Identify basic air cooling technology.**
2. **Learn thermodynamic rules governing heat flow into and out of fluids like air and water.**
3. **Identify options for bulk air heating and cooling installations based on need and application.**
4. **Calculate the amount of air heating or cooling needed.**
5. **Explain measures of heating and cooling system performance.**

Get Started

Go to the Course Index

About the Aeolus Project

## Course Summary:

| Date | Details |
|---|---|
| | Bulk Air Heating and Cooling (https://canvas.instructure.com/courses/1175124/assignments/6497257) |

# 1.0 Introduction

The equipment and infrastructure required to heat or cool the large quantities of air that are required in mining applications are typically massive, complex, and expensive. While the goal of this course is to familiarize the user with the basic theory and components of such systems, the design, manufacture, installation and maintenance of these systems should only be performed by expert-level practitioners with sufficient training and experience.

Although air heating and cooling can be quite complex in practice, consisting of linked installations of compressors, refrigerant, heat exchangers and kilometers of piping; the thermodynamic principles that govern the flow of heat into, or out of an air stream are actually quite simple.

Whether the desired effect is to add heat to air, or remove heat from air, only a few thermodynamic principles and equations are required to understand the behavior of heat as it flows into and out of an airstream.

The heat that flows into, or out of a fluid is governed by the following equation:

**Heat Flow Equation:** $q = mC_p\Delta T$

where:

$q$ = heat (kW)

$m$ = mass flow rate of fluid (kg/s)

$C_p$ = Specific Heat of fluid - air, water, refrigerant (kJ/kgK)

$\Delta T$ = change in fluid temperature (K)

Two of the most common Specific Heat values encountered in mine air heating and cooling calculations are:

Water = Cp of 4.187 kJ/kgK

Air = Cp of 1.005 kJ/kgK

If required, the Cp of other common refrigerants can be looked up in tables that can be found online.

# 2.0 Sources of Heat in Mines

There are many sources of heat in mines, including natural and man-made.  Common sources of mine heat include:

1. Auto-compression of ventilating air
2. Blasting
3. Conveyors
4. Diesel Equipment
5. Electrical Equipment
6. Rock Strata
7. Water inflow

Additional information regarding the sources of mine heat, and the physiological effects of heat on mine workers may be found in the "Metal/Nonmetal Mine Ventilation System Design" course attached to this series of modules.

# 3.0 Refrigeration and Cooling

Refrigeration, or cooling of air or other substances is accomplished by removing heat, or energy from whatever it is that is being “cooled”.  This heat energy must then be rejected/absorbed from the system such that the first law of thermodynamics can be preserved.  Often, a large body or reservoir of air or water is used as a “heat sink”, or repository for waste heat removal.  It is important to remember when calculating heat loads in mines, that any heat that will be removed from the mine atmosphere must be also rejected from the system in such a way that the desired cooling effect can be maintained.

In order to accomplish this, a refrigerating machine must raise the temperature of its refrigerant to a temperature higher than the greatest temperature reached by the temperature of the environment.  This requires the input of mechanical work, or in some cases heat energy.  The most common refrigerators, or vapor-compression machines convert mechanical energy in the form of compressors, which are used to raise the temperature and pressure of a low-temperature, low-pressure vapor, which is then condensed into a high-temperature, high-pressure liquid.  This liquid is then passed through an expansion valve (nozzle) where it becomes a low-temperature, low-pressure liquid.  As this liquid passes through an evaporator, heat from the ambient air is absorbed, effectively cooling the air passing over the evaporator as the low-temperature, low-pressure liquid is transformed back into a low-temperature, low-pressure vapor.  These phase changes in the refrigerant occur cyclically in a closed-loop system driven by the input of mechanical power to the compressor(s) and in many cases, fan(s).  Heat is absorbed by the refrigerant from one environment as it evaporates, and is rejected into another environment as it condenses.

Figure 3.1 shows a basic vapor-compression cycle with heat absorbed by the evaporator and rejected at the condenser.

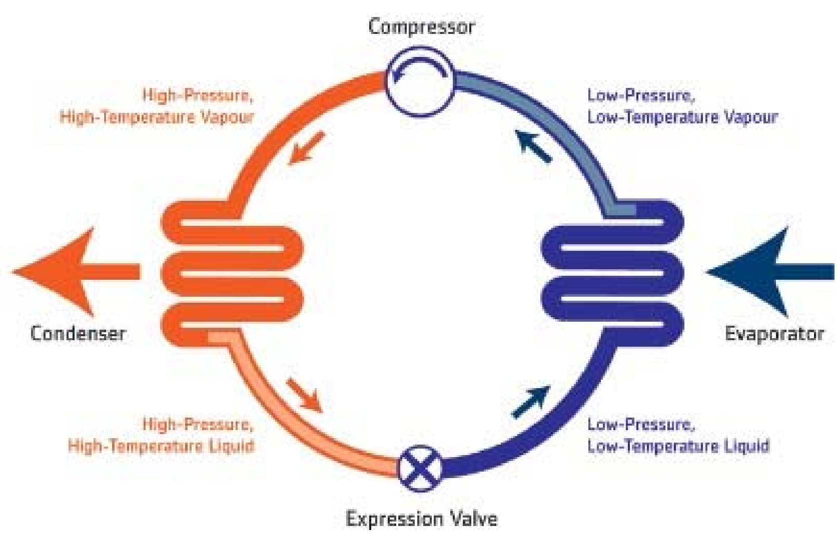


**Figure 3.1: The Vapor Compression Cycle.**

# 3.1 The Carnot Cycle

In 1824, Nicolas Carnot published a theory explaining the behavior of heat engines including the “ideal” thermal engine behavior, which has since become known as the Carnot Cycle.  The Carnot Cycle consists of four reversible steps or processes:

1. Isothermal (constant temperature) compression, producing heat ($q_{out}$)
2. Adiabatic (constant heat) compression
3. Isothermal expansion, requiring heat ($q_{in}$)
4. Adiabatic expansion

The Carnot Cycle and a graph of Pressure versus Enthalpy diagram is shown on Figure 3.2.

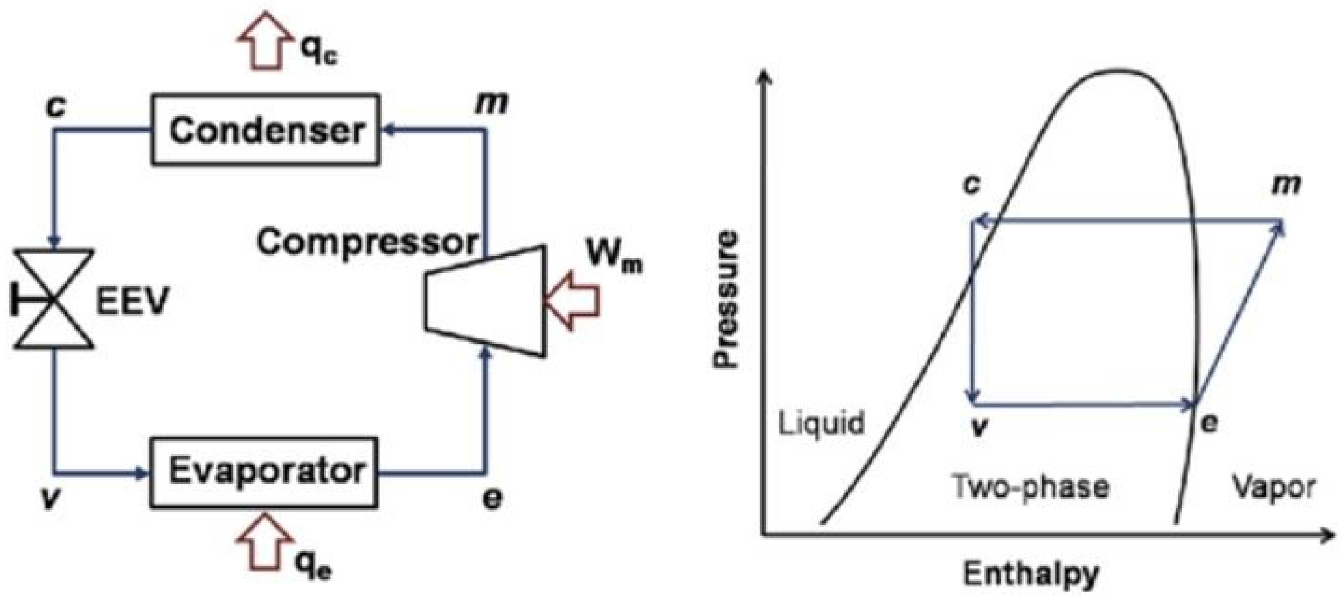


**Figure 3.2: Ideal Carnot Cycle and accompanying P vs H diagram.**

An actual Pressure versus Enthalpy diagram for the common refrigerant R134a is shown on Figure 3.3.

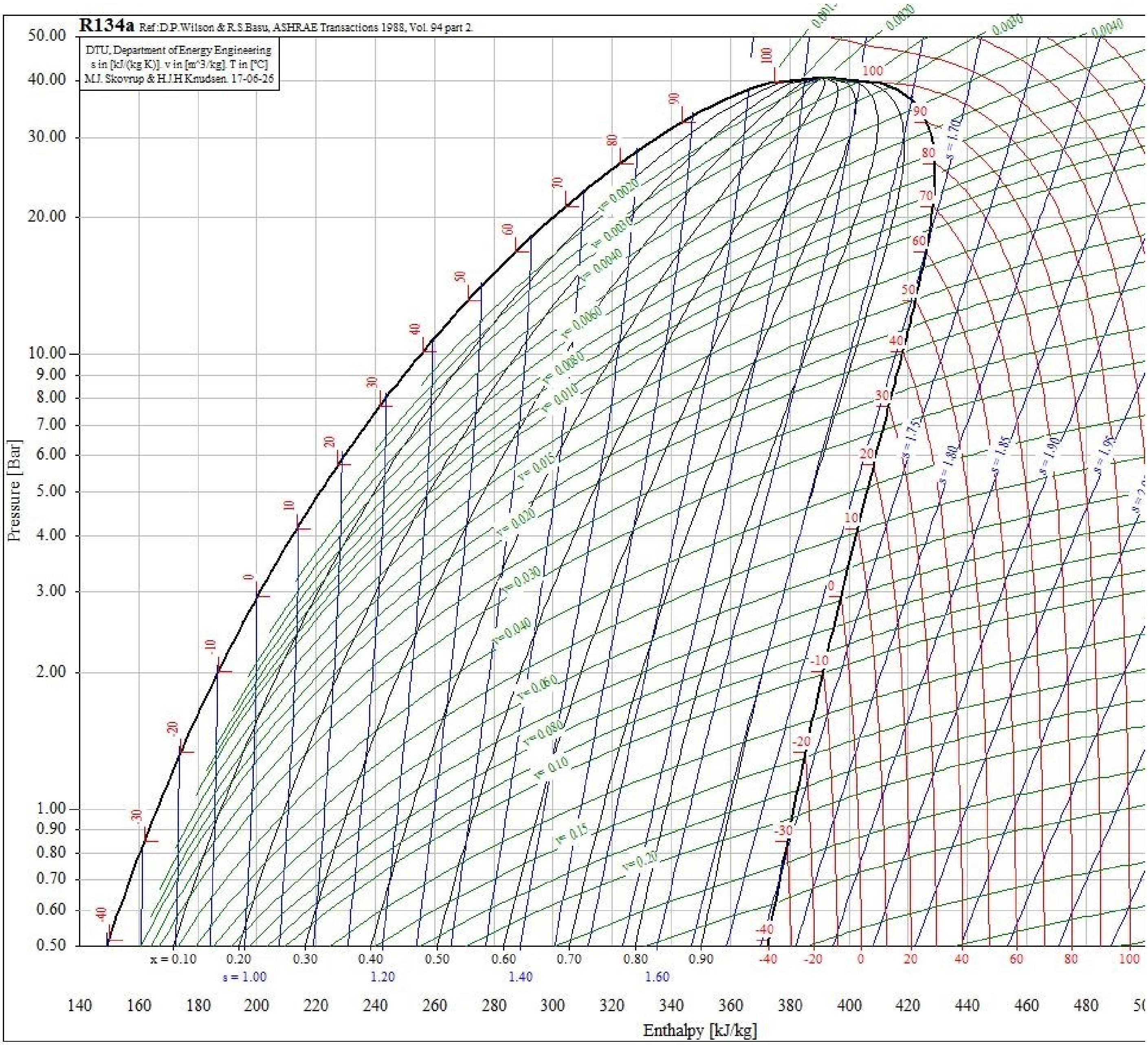


**Figure 3.3. Pressure vs. Enthalpy diagram for R134a (ASHRAE).**

# 3.2 Coefficient of Performance (COP)

The thermodynamic efficiency for an ideal Carnot Engine is given by the coefficient of performance (COP), defined as the ratio of the Temperature of the refrigerant at heat extraction to the difference in refrigerant Temperature between heat extraction and rejection:

Coefficient of Performance for Ideal Carnot Engine:

$$\textbf{Coefficient of Performance for Ideal Carnot Engine:}\ COP = \frac{T_{ext}}{T_{rej} - T_{ext}}$$

where:

$COP$ = Coefficient of Performance

$T_{ext}$ = Temperature of refrigerant during heat extraction (C)

$T_{rej}$ = Temperature of refrigerant during heat rejection (C)

The thermodynamic efficiency for an ideal vapor-compression refrigerator for example, is governed by the relative saturation temperatures of evaporation and condensation for the refrigerant used. In this case (for a vapor-compression machine) COP can be defined as:

Coefficient of Performance for an Ideal VP Machine:

$$\textbf{Coefficient of Performance for an Ideal VP Machine:}\ COP = \frac{T_{evap}}{T_{con} - T_{evap}}$$

where:

$COP$ = Coefficient of Performance

$T_{evap}$ = Saturation Temperature of refrigerant during evaporation (C)

$T_{con}$ = Saturation Temperature of refrigerant during condensation (C)

However, the Carnot Cycle represents ideal behavior only- in reality the process is affected by frictional and thermal losses that reduce the efficiency of the system. In reality, the COP for a given vapor-compression machine is governed by the ratio of its cooling capacity (heat extraction) and the mechanical power required.

Coefficient of Performance for an Actual VP Machine:

$$\textbf{Coefficient of Performance for an Actual VP Machine:}\ COP = \frac{HE}{MW}$$

where:

$COP$ = Coefficient of Performance

$HE$ = Heat Extraction (W)

$MW$ = Input Mechanical Work (W)

Figure 3.4 shows the same P vs h diagram for R134a with the refrigeration cycle highlighted (red).

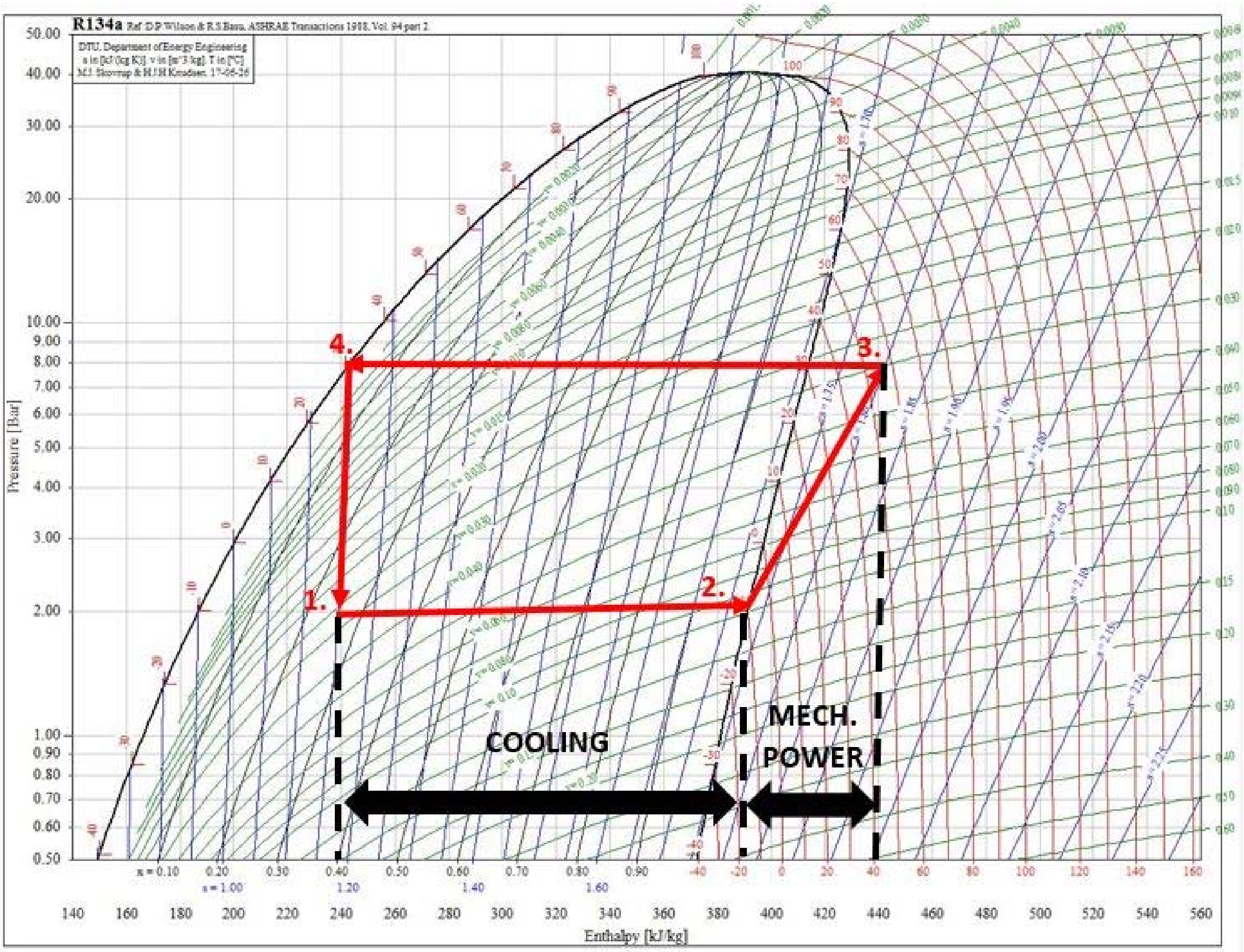


**Figure 3.4: Pressure vs. Enthalpy diagram for R134a with refrigeration cycle highlighted (ASHRAE).**

The heat extracted by the system, or its cooling power is determined based upon the mass flow rate of the refrigerant and the change in enthalpy:

where: $$q_{HE} = m\frac{H_2 - H_1}{MW}$$

$q_{HE}$ = Heat Extracted by the machine (kW)

$m$ = mass flow rate of refrigerant (kg/s)

$H$ = Enthalpy of refrigerant (kJ/kg)

Typically, COPs for large surface air chillers range from 5 to 6, while an underground chiller may only provide a COP of 2 to 4.

# 4.0 Industrial Refrigeration Equipment

Bulk air cooling equipment, including a variety of water chillers and ice makers for mining applications are large, complex, and expensive installations that require substantial engineering. In practice, cooling systems may include multiple circuits of refrigerant, multi-stage and/or variable geometry compressors and many other features that improve efficiency while adding complexity (and cost). However, no matter their size or complexity, at the heart of all major industrial refrigeration equipment lies a compressor, condenser, expansion valve and evaporator. But in order to cool mine intake air, and reject heat into the atmosphere, refrigeration plants must also have a minimum of two heat exchange systems- one to extract heat from the mine inlet airstream and one to reject that heat. Most commonly, this heat absorption and rejection are done through the use of water, whereby chilled water is supplied to absorb heat from the intake air and then transmit that heat to the ambient environment.

Figure 4.1 shows the footprint of a typical surface mine air cooling facility.

**Figure 4.1: Modern, mine refrigeration plant including cooling towers. (Mining Technology).**

# 4.1 Water Chillers

Water chillers are used to cool water for use in spray chambers.  They generally include a vapor-compression chiller with a heat exchanger that utilizes two, closed-loop water circuits.  The chilled water circuit consists of water that is cycled to the spray chamber/bulk air cooler on surface (or sent underground in some cases) while the warm-water circuit moves water back and forth from the cooling tower.  These chillers are modular in nature, and can be scaled up and installed in parallel as necessary to supply the required amount of cooling.

The most common type of water chiller utilizes a centrifugal compressor (may be single-stage or multi-stage).

Figure 4.2 shows the components of a typical R134a water chiller.  Figure 4.3 depicts an example of an actual, commercially available 400-ton R134a chiller.

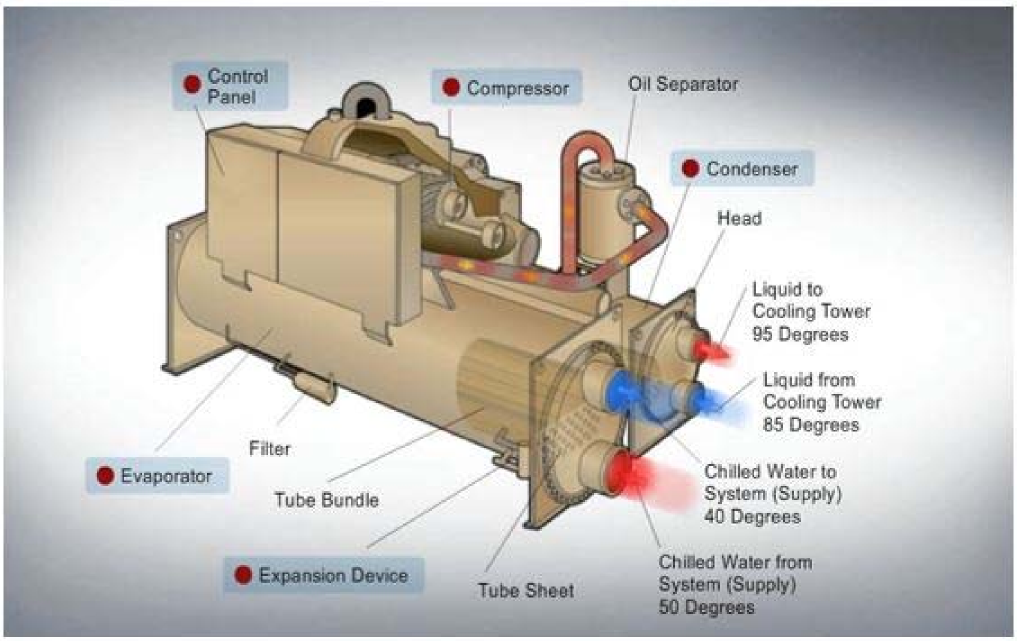


**Figure 4.2: R134a Chiller schematic.**

**Figure 4.3: 400-ton Chiller (Carrier).**

The heat exchangers utilized on commercial chillers are most commonly shell and tube type, or shell and plate type.

Shell and tube exchangers force refrigerant through a series of small diameter tubes inside a tubular case or “shell”.  Fluid such as water or glycol is then forced around the tubes and controlled by a series of baffles designed to extend the residence/contact time.

Figures 4.4 and 4.5 show a schematic view and an actual Shell and Tube heat exchanger, respectively.

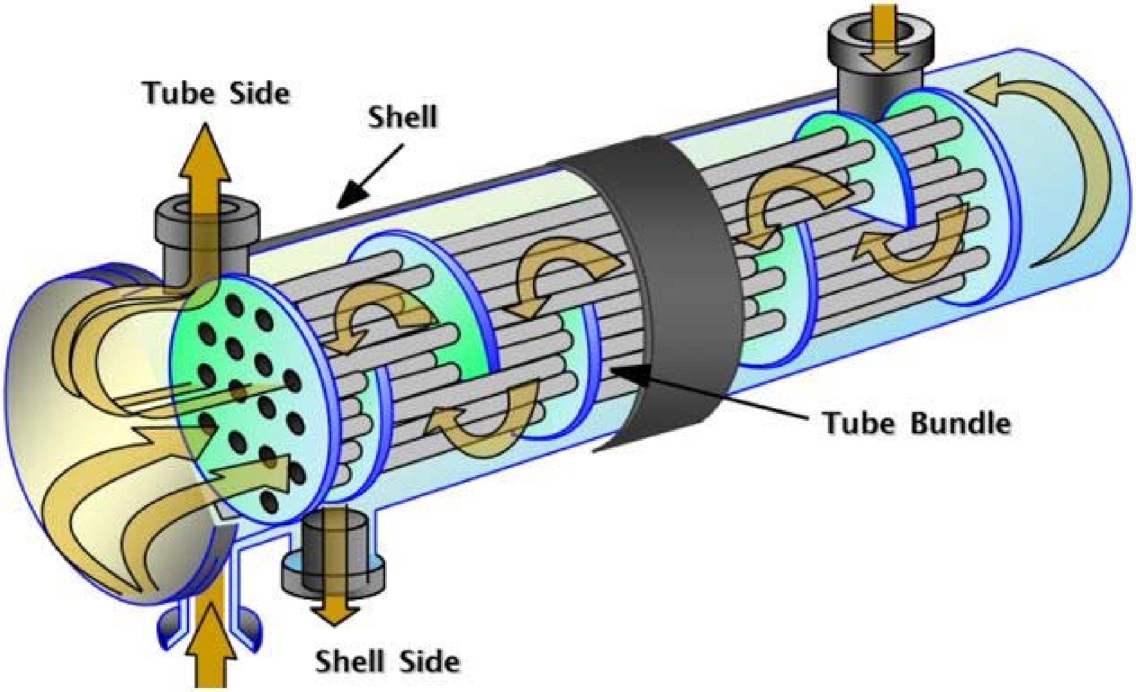


**Figure 4.4: Shell and Tube heat exchanger (Schematic View).**

**Figure 4.5: Shell and Tube heat exchanger (Actual).**

The process for a Plate and Frame type is similar, with the two fluids forced over alternating metal plates that have channels milled into their surface to assist in heat transfer. Inlet and return flows are from the same side of the exchanger, as the two fluids enter and exit the unit in parallel.

Figures 4.6 and 4.7 show a schematic view and an actual Plate and Frame heat exchanger, respectively.

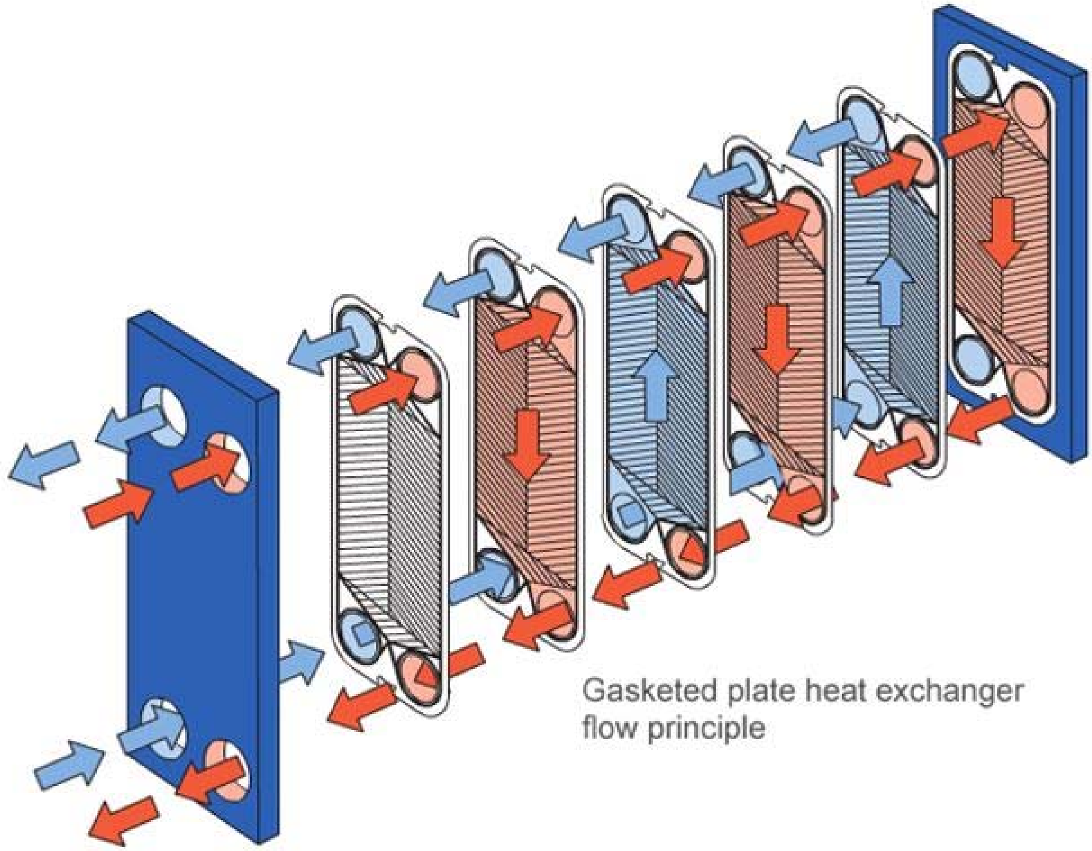


**Figure 4.6: Plate and Frame heat exchanger (Schematic View).**

**Figure 4.7: Plate and Frame heat exchanger (Actual).**

Ultimately, the choice of which type of heat exchanger to use will likely be made by the manufacturer of the chiller or plant, but a sound understanding of the components of such systems is nonetheless valuable to the ventilation engineer.

An alternative to water chillers that utilize centrifugal compressors can be specified to include large, screw-type compressors. The capabilities of these screw compressors, which unlike centrifugal compressors are positive-displacement devices, are generally much greater, leading to large complex installations and greater refrigerant loads. Most such installations can be found on surface, and utilize ammonia as the coolant.

Screw-compressor chillers also differ from centrifugal compressor chillers in one other important aspect; they are fully immersed in oil that cools and lubricates them. This oil must be recovered in an oil separator and cooled separately. Thus the energy required to cool this oil may be as much as one-third of the total compressor input power. As a result, the COP for an ammonia (i.e., screw-type) chiller should be adjusted to account for this additional power requirement.

Figure 4.8 shows a schematic of an ammonia plant (screw-type compressor) used to provide chilled water for a bulk air cooling installation. Figure 4.9 shows an actual ammonia chiller installation.

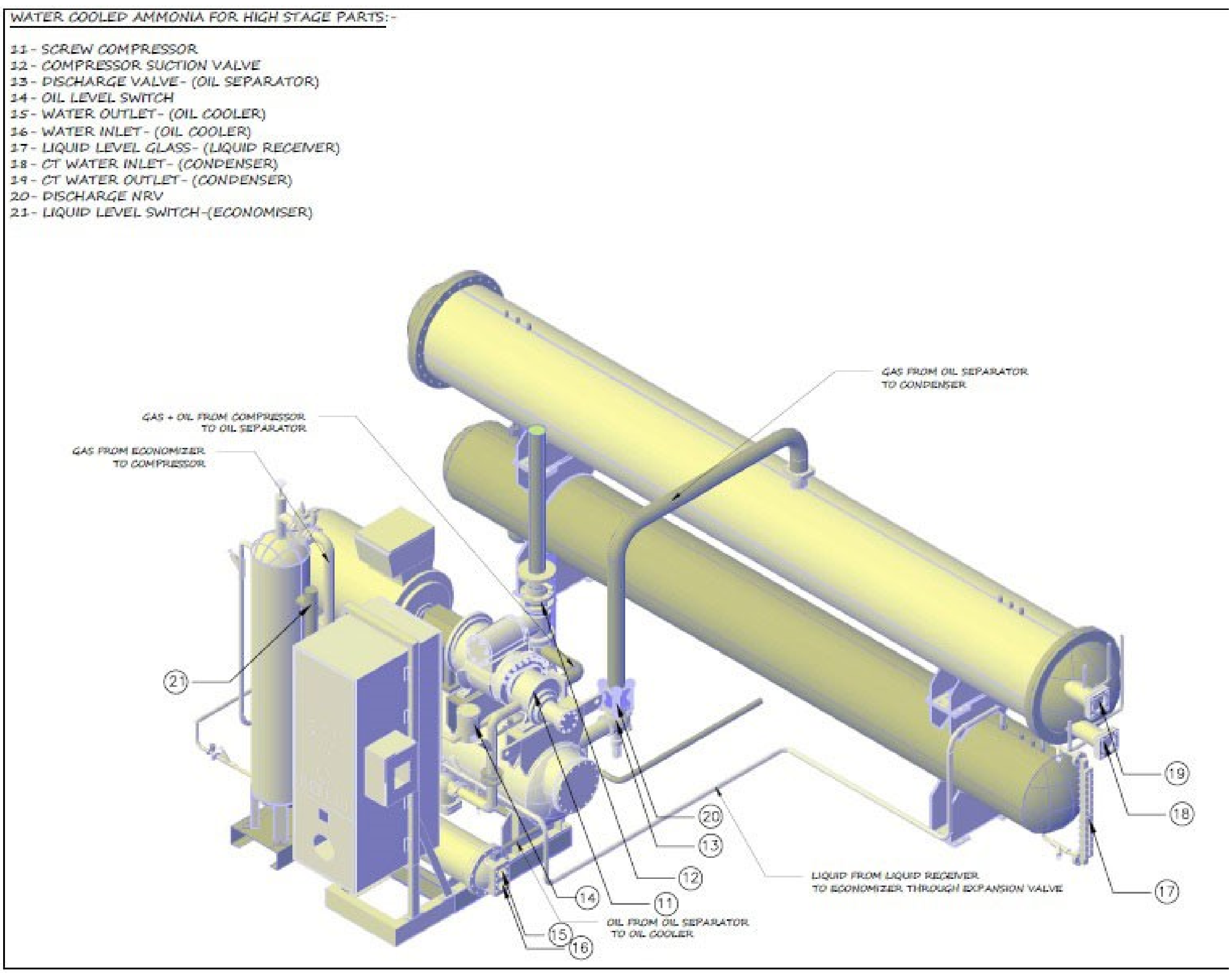

Figure 4.8: Schematic of a screw-type compressor using ammonia refrigerant.

Figure 4.9: Ammonia plant with screw-type compressor.

# 4.2 Ice Makers

Ice makers cool water until it freezes, forming hard ice or an ice/water mixture sometimes referred to as slurry ice. The ice may then be used to cool water or brine that is sent to an air cooler, preserved as part of a thermal storage program, or sent underground in hard or soft form depending on the design of the cooling system. Ice can be especially effective in underground cooling applications due to the latent heat of fusion (required to induce a phase change from the solid form of water to the liquid).

So called "hard" ice, or what traditionally might be thought of as ice (cubes, chips, flakes, etc.) is typically made as water flows over large metal plates that are cooled to sub-freezing temperatures by the circulating refrigerant. As additional water flows over the plates (and eventually ice) the ice layer gradually thickens. Periodically, this coolant flow is stopped, and in some cases reversed, such that a surges through the plates, melting the layer of ice closest to the plates and causing the plates to "shed" their ice for collection. This process is repeated continually. Since the process is cyclical, and does not have either a continuous output or power draw, these hard-ice machines are often installed in parallel and off-set so that the timing of their cooling and harvesting cycles equate to a near continuous output.

"Soft" ice, or slurry ice is made in a continuous process by steam-injection or vapor-compression, and provides an extremely consistent ice and water mixture that is pumpable for easy transport.

Recalling that heat flow is governed by the equation:

$$q = mC_p\Delta T$$

Consider the difference in the quantity of water required to provide 30 MW of cooling between chilled water (delivered at 5 degrees C) and ice (delivered at -5 degrees C). Assume that the return water stream temperature is 22 degrees C.

For water, it is possible to use the simplified heat flow equation:

$$q = mC_p\Delta T$$

30,000 = m(4.187(22-5))

<u>m = 421 kg/s</u>

For Ice, it is necessary to calculate the heat flow required to raise the temperature to 0 degrees C…

q = m(2.050(0-(-5)))

q = m(10.25) kW

…as well as the latent heat of fusion required to melt the ice (per kg):

q = m(334) kW

…in addition to the heat flow out of the liquid water:

$$q = mC_p\Delta T$$

30,000 = m(4.187(22-0))

q = m(92.1) kW

Which gives a final mass flow of water to be:

30,000 = m(10.25+334+92.1)

m = 69 kg/s

By taking advantage of the latent heat of fusion (heat required to melt ice), it is possible to reduce the mass of water required to deliver 30 MW of cooling by 84%!

Although this represents a significant savings in transportation and pumping costs particularly when the cooling is needed in very deep mines, there is a penalty to pay in terms of infrastructure cost and maintenance. This is because ice making machines typically cost more to purchase, install and maintain than simple water chillers. Depending on the economics of the individual project, however; ice makers may provide a significant benefit, especially in cases where extreme heat or shaft depth are encountered. Typically, ice is only used in cases where mine depths exceed 7,200 ft.

McPherson recommends that the following four points be carefully considered for any potential ice making system:

1. The scale of ice making
2. The method of transport
3. How it will be used to cool the mine environment
4. The cost of the system

## Case Study: Mponeng Mine

Mponeng Mine is currently the world's deepest operating mine. Located in the Witwatersrand of South Africa, this operating gold mine currently reaches depths of almost 13,000 ft below the surface. Gold mining in the Witwatersrand is labor-intensive enterprise, requiring the presence of thousands of miners underground during each shift at Mponeng. The mine has a dedicated ventilation staff of 16 employees, who perform air quality checks daily, and manage a full airflow quantity survey approximately once per quarter.

Virgin Rock Temperatures in the mine can reach as high as 155 degrees Fahrenheit. In addition to the heat of the rock strata, heat is added to the mine environment by diesel equipment, blasting, conveying equipment and groundwater. The single largest source of heat in the mine is due to auto-compression of the air as it travels from the surface down to the active levels of the mine.

Auto-compression currently accounts for approximately 41 Megawatts of the 70 Megawatts of cooling provided by the mine ventilation system. In total, the mine has 140 Megawatts of cooling capacity installed, split between bulk air coolers, chilled water, hard ice and soft ice (slurry ice). In order to effectively transport the required 180 tons per hour of slurry ice to the shaft, a flexible pipe conveyor is used. Mine air is cooled to approximately 84 degrees Fahrenheit at the working faces in order to protect the safety and health of the workers underground.

The formation of ice on surface that is then transported underground is a relatively inefficient and expensive process that can only be justified in the world's deepest and hottest mines.

# 5.0 Bulk Air Cooling

In order to cool the air entering the mine, it is necessary to extract heat from the air, usually through direct contact with chilled water.  Keeping in mind the scale of most mine ventilation systems, it is clear that this is an undertaking that requires significant planning and infrastructure construction.

Figure 5.1 provides a schematic of the cooling system components for a typical bulk air cooling installation including the chiller(s), spray chamber and cooling tower(s).

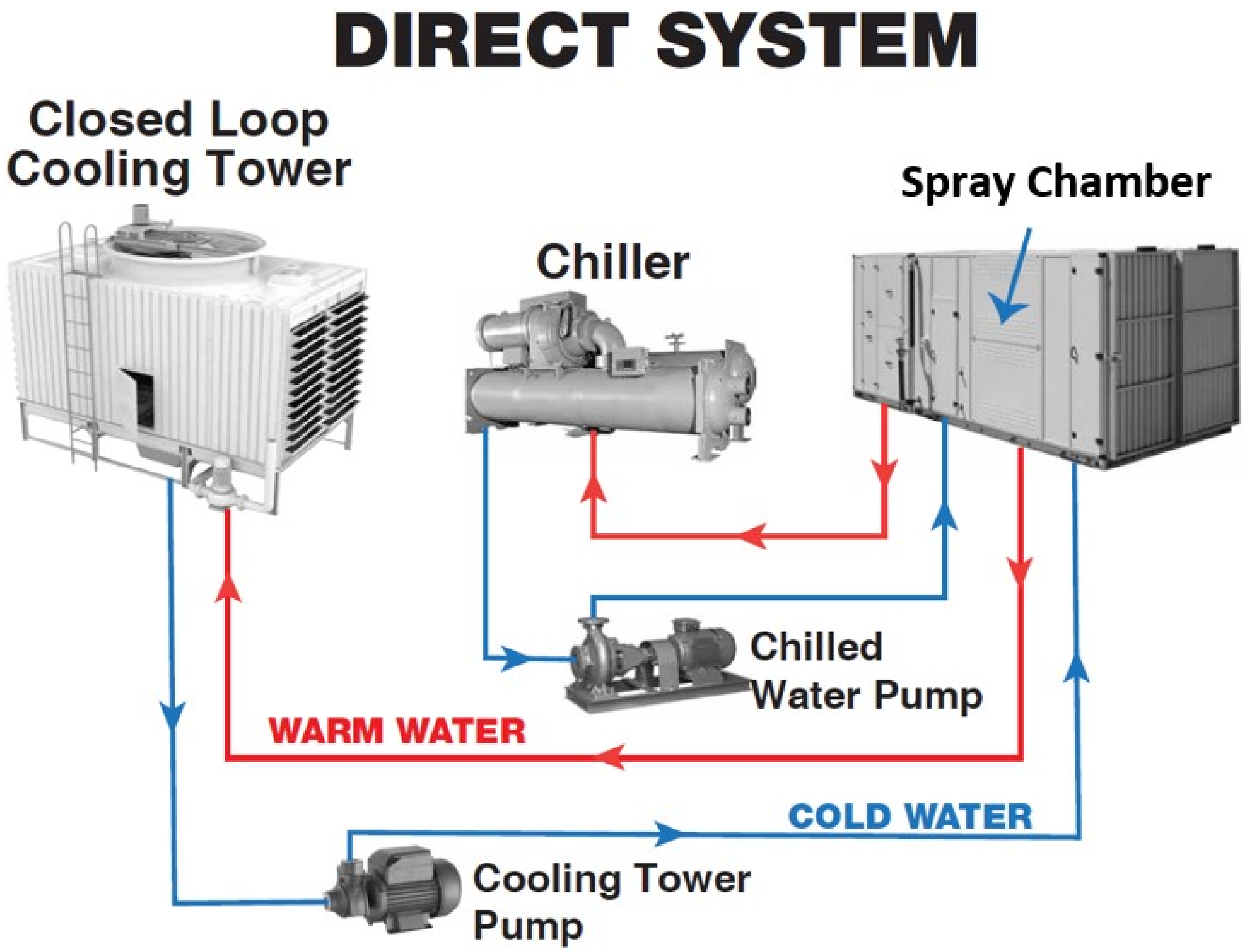


**Figure 5.1: Schematic of a typical Bulk Air Cooling Installation.**

# 5.1 Spray Chambers

In the context of bulk air cooling, the spray chamber acts as a water/air heat exchanger, whereby heat from the ambient air is extracted through direct contact with cool water from the chiller(s). Air is drawn through the entrance to the chamber (or pushed directly from inlet fans) and through a series of water sprays that are directed across (perpendicular) or against the direction of airflow. Heat from the air is absorbed by the water, causing both the temperature and humidity of the air to drop. Finally, the cooled air is directed into the shaft (in the case of a surface installation) or drift and the warmed water collects in a sump in the bottom of the chamber and is pumped out in an enclosed pipe. In a closed-loop system, the wastewater (warm) is sent back to another heat-exhanger (e.g., cooling tower) for re-use.

Figure 5.2 shows a simplified layout of a typical horizontal spray chamber.

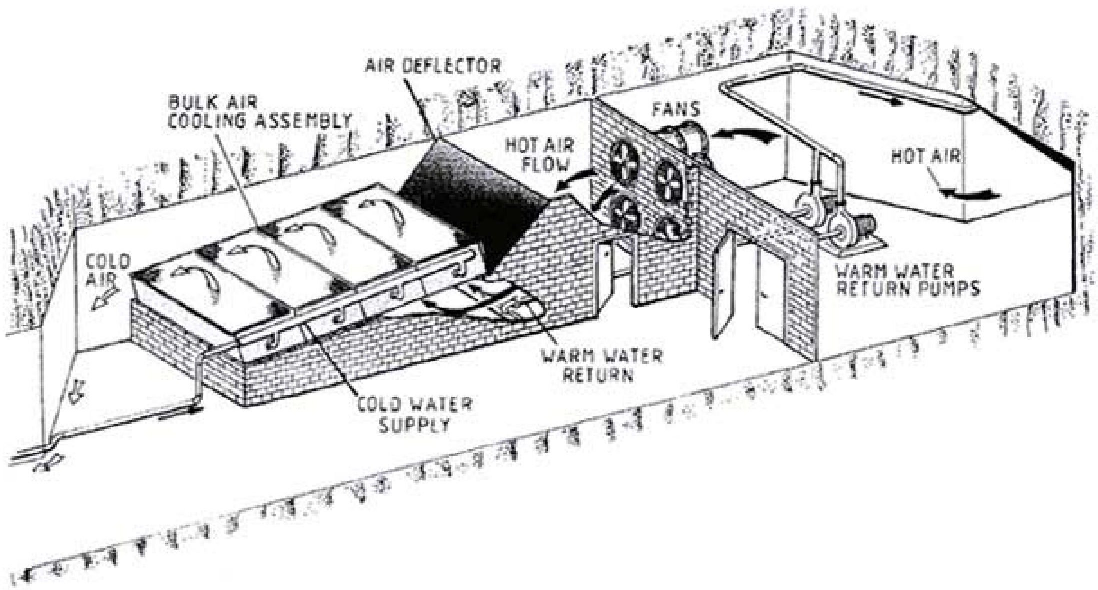


**Figure 5.2: Spray chamber layout (Heat and Fluid Engineering).**

Figure 5.3. shows the interior view of a spray chamber including the water sprays/nozzles.

**Figure 5.3: Inside view of a spray chamber (BBE).**

Figure 5.4 shows an exterior view of the spray chamber and inlet fans at Impala Platinum's Shaft No. 11C near Rustenburg, South Africa.

**Figure 5.4: View of the spray chamber and fans at Impala Shaft No. 11C.**

The efficiency or effectiveness of the process is heavily dependent on the following factors:

1. Mass flow-rate of water
2. Mass flow-rate of air
3. Temperature of cooled water (sprays)
4. Temperature and humidity of ambient air (inlet conditions)
5. Residence (contact) time

Of these factors, only the temperature and humidity of the ambient conditions lies beyond the control of the design engineer. The flow rates of water and air will be controlled by the selection of pumps and fans, respectively. Water spray temperature can be controlled through the design of the chiller. The residence time is function of the spray chamber geometry (length, width, height) adjusted to provide the desired air velocity for the prescribed flow-rate. The optimal velocity for horizontal spray chambers lies between 500 and 1,500 fpm (Burrows, 1982). Both the airflow and the water sprays should be evenly distributed across the spray chamber, with an optimum water spray density between 3 and 8 gpm per square foot (Bluhm, 1983).

# 5.2 Cooling Towers

In a typical bulk air cooling installation, the spray chamber is utilized to extract heat from the mine intake air, while the cooling tower is used to reject heat from the water stream prior to being sent to the chiller; however, the performance characteristics of a cooling tower are essentially the same as for a spray chamber. In fact, the only difference for our purposes is that the "tower" is oriented vertically, while the spray chamber is considered to be a horizontal installation. Ambient air enters the bottom of the tower and is drawn upwards through cascading water falling into a sump at the bottom of the tower before passing through a baffle or "drift eliminator" and passing out of the top of the tower through a large-diameter low-pressure, high volume fan.

Figure 5.5 shows a simplified cooling tower schematic.

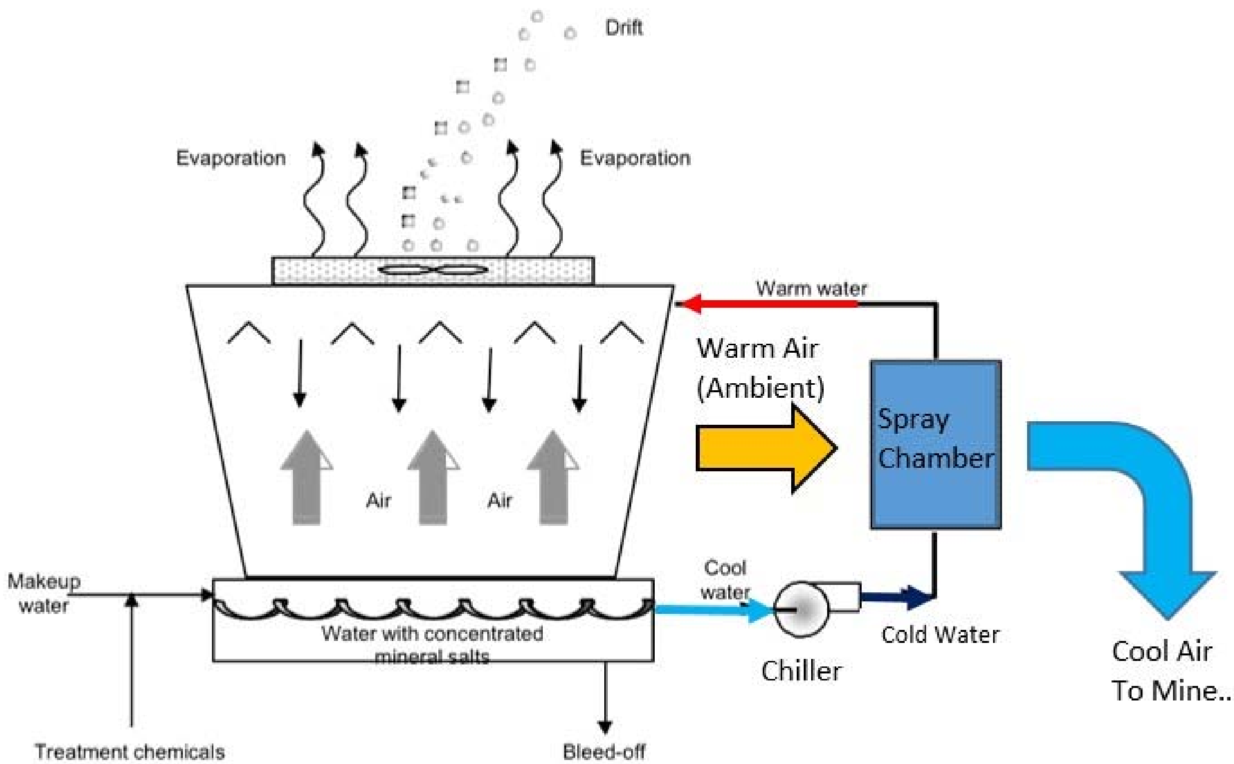


**Figure 5.5: Schematic of typical cross-flow type cooling tower (no packing).**

Figure 5.6 shows the surface cooling towers at Impala Platinum's Shaft No. 11C near Rustenburg, South Africa.

**Figure 5.6: Surface cooling towers at Impala Shaft No. 11C.**

# 5.3 Cooling System Performance

While several methods have been suggested for quantifying the performance of spray chambers and cooling towers, the most advanced has been developed by Whillier in 1977 and improved upon by Bluhm in 1984. This work has led to the “Factor of Merit”, which has proven to be the most successful measure of direct air-to-water heat exchanger performance to date. According to the First Law of Thermodynamics, the change in energy (heat) of the air must equal the change in energy of the air. The Second Law dictates that the water cannot escape the tower with a wet-bulb temperature lower than that of the air that enters; and that similarly it is not possible for the exiting air to have a wet-bulb temperature greater than that of the inlet water. The reverse is also true for a spray chamber: the temperature of the air exiting the chamber cannot exit with a temperature lower than that of the water sprays; it is not possible for the water exiting the sump to do so with a temperature higher than the inlet air. Note that dry-bulb temperatures have no effect on the process at any stage. Empirical study of cooling tower performance has led to the development of a Factor of Merit for various types of cooling towers, similar to the way in which the performance of heat exchangers is given a U-factor. Unfortunately, the calculation of Factors of Merit is complex and time-consuming. Fortunately, it is possible to make a reasonable assumption based on the empirically developed values.

Table 5.1 gives the Factors of Merit for mechanical draft cooling tower and horizontal spray chamber configurations developed by J.H.J. Burrows.

<table>
<tr><th colspan="2">Type of Heat Exchanger</th><th>Factor of Merit (F)</th></tr>
<tr><td rowspan="3">Sprays located in a horizontal tunnel with no packing</td><td>1 Stage</td><td>0.40 to 0.55</td></tr>
<tr><td>2 Stage</td><td>0.58 to 0.67</td></tr>
<tr><td>3 Stage</td><td>0.67 to 0.75</td></tr>
<tr><td rowspan="2">Vertical spray-filled tower, no packing</td><td>High water loading</td><td>0.50 to 0.60</td></tr>
<tr><td>Low water loading</td><td>0.60 to 0.70</td></tr>
<tr><td rowspan="2">Vertical, with packing</td><td>High water loading</td><td>0.55 to 0.65</td></tr>
<tr><td>Low water loading</td><td>0.65 to 0.75</td></tr>
<tr><td colspan="2">Industrial packed towers</td><td>0.60 to 0.70</td></tr>
<tr><td colspan="2">Industrial densely-packed towers</td><td>0.70 to 0.80</td></tr>
</table>

**Table 5.1: Factors of Merit for various cooling towers and spray chambers (Burrows, 1982).**

# 6.0 Air Heating

In many mining environments, the ambient air temperature falls below freezing for significant periods of time. Although this may not always be a problem, prolonged exposure to sub-freezing temperatures can cause both health and safety concerns for mine workers, and may also negatively impact the stability, production and equipment of the mine. For all of these reasons, the heating of mine intake air may sometimes be required.

In theory, if not in practice, the heating of mine ventilation air is easier to accomplish than cooling. The thermodynamics involved are actually quite simple.

The complications and difficulties surrounding bulk air heating are associated with the methods of applying heat to the air, and the cost and maintenance of the heating infrastructure themselves, as well as the choice of fuel. Many options exist for bulk air heating in mining applications, including direct (e.g., propane burners) and indirect methods (e.g., glycol with plate heat exchangers) as well as a host of alternatives (e.g., ice stopes, waste heat recovery, etc.).

The determination of how much heating is required is made based upon the expected minimum temperature of the environment and the minimum allowable reject temperature to protect the mine assets (e.g., personnel, equipment, and infrastructure). The total heat required can then be calculated with the Heat Flow Equation:

**Heat Flow Equation:** $q = mC_p\Delta T$

where:

$q$ = heat (kW)

$m$ = mass flow rate of air (kg/s)

$C_p$ = Specific Heat of air, 1.005 (kJ/kgK)

$\Delta T$ = change in air temperature (C/K)

**Example:**

Winter temperatures in Alaska can reach -50 degrees C. In order to protect the mine personnel and services, a certain mining company requires that intake air have a minimum temperature of +5 degrees C. The mine requires 500,000 cfm of airflow, at a density of 0.075 lb/ft$^3$. How much heat is required (in Megawatts) to meet the minimum temperature requirement?

$$q = mC_p\Delta T$$

m = (500,000 cfm / 2110 cfm per m$^3$/s) X 1.20 kg/m$^3$ = 284 kg/s

Cp = 1.005 kJ/kgK

$\Delta T$ = (5 – (-50)) = 55 degrees C

q = 284 X 1.005 X 55 = 15.7 MW

# 6.1 Direct Heating

The direct heating of the mine intake air is the most efficient means of increasing the air temperature entering the mine.  Typically, a fuel such as propane or natual gas is pumped into a matrix of burners evenly distributed across the airstream.  The air passes directly over the flames, ensuring excellent heat transfer.  In this method of air heating, the products of combustion are passed directly into the ventilating air, where they are quickly diluted.  Nonetheless, the potential for contamination from CO does exist, and the system should include an interlinked series of CO monitors downstream from the burners that can immediately shut down the heater should elevated levels be detected.  Proper inspection and maintenance of the system can prevent the incomplete combustion that leads to elevated CO production.

Figure 6.1 shows a typical direct-fire mine air intake heater.

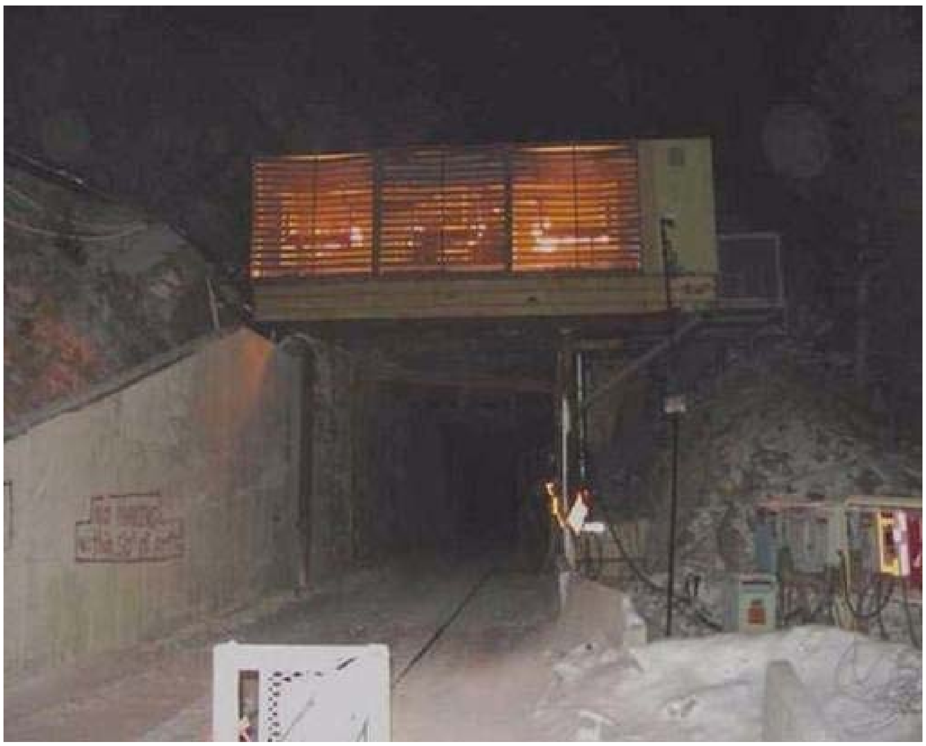

**Figure 6.1: Typical direct-fired air heater mounted over the intake portal.**

# 6.2 Indirect Heating

Indirect heaters work by burning fuel in a boiler that heats a water/glycol mixture that can then be pumped into a heat exchanger located in the airway.  This eliminates the potential contamination from any products of combustion in the intake airstream, but does come at a cost of some thermal efficiency.  For some, the additional safeguards and flexibility that an indirect heating system offers are worth the additional expense.

### Case Study: Rio Blanco Mine (Andina)

Rio Blanco Mine is a copper block-caving operation located between 9,000 and 12,000 ft above sea level in the Andes Mountains of Chile.  In support of a mine expansion project, an system of indirect, diesel-fueled mine air heaters was installed on the surface at an elevation of almost 13,000 ft.

Due to concerns about having fuel and open flames in the main mine intake (with capacity of over 1,500,000 cfm) an indirect air heating installation was specified.  In order to provide the safest possible installation, diesel fuel is stored remotely, then pumped to high-efficiency boilers that utilize a closed-loop glycol system to heat mine air through a series of plate-type heat exchangers.

Further safeguards against the potential for a fire include fire doors capable of completely isolating all areas that contain fuel from those that supply the mine with fresh air and an interconnected fire monitoring and extinguishing system (water sprinklers).

As of 2016, the air heaters had been operating for more than six years without problems, and have been proven to consume less fuel than was anticipated.  This installation represents one of the largest known, and safest mine air heating installations in the world.

# 6.3 Alternative Methods of Heating

In addition to combustion-type air heaters, many other options exist for the heating of mine intake air, including the recovery of waste heat from the mine exhaust or other on-site industrial processes, the use of geothermal heat sources and the use of “ice stopes”.

In the bulk air cooling section of this module, we have already discussed the incredible amount of heat required to melt or freeze water/ice (latent heat of fusion). Just as this was used to great advantage in extracting heat from mine air (by melting frozen ice), the reverse can be just as effective in heating mine air (by freezing water) if suitable conditions exist. Ice stopes require relatively large excavated caverns to be located somewhere along the intake pathway in between the surface and the working areas of the mine. High-pressure nozzles are used to blow fine droplets of water into the sub-freezing air, resulting in a “snowfall” underground. As the water undergoes a phase change, this time from liquid to solid, the latent heat is transferred from the water to the air, warming it as it passes through the stope. Some heat from the liquid water is transmitted to the air directly, although the latent heat transfer is most significant.

Figure 6.2 shows a diagram explaining the operation of an ice stope.

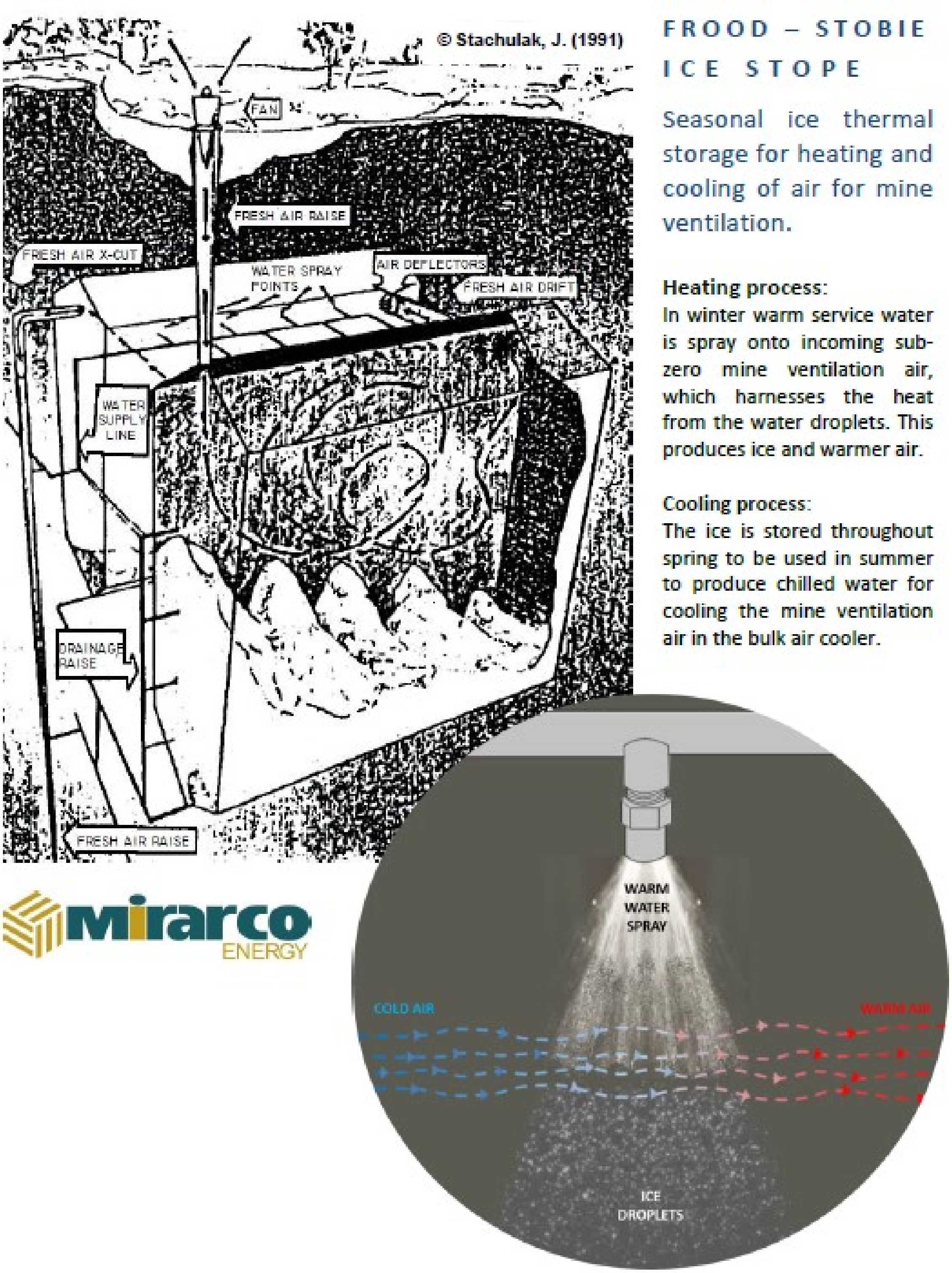


**Figure 6.2: Ice stope operation at Vale's Stobie-Frood Mine (MIRARCO).**

Although this application is quite rare, owing to the specialized conditions that it requires (e.g., large excavated openings along the intake circuit), the process is capable of providing significant heat at a relatively low cost.  It also has the added benefit of reversibility- in the warm Summer months, the melting ice provides significant cooling to the mine- all for free!

**Example:**

**Heat Flow Equation:** $q = mC_p\Delta T$

where:

$q$ = heat (kW)

$m$ = mass flow rate of air (kg/s)

$C_p$ = Specific Heat of air, 1.005 (kJ/kgK)

$\Delta T$ = change in air temperature (C/K)

**Example:**

In order to heat the mine air, a company sprays 50 liters per second of water at 10 degrees C into an ice stope. The inlet air temperature is -20 degrees C. How much air can the stope heat up to the mine reject temperature of 5 degrees C?

For water: $q = mC_p\Delta T$

m = 30 kg/s

Cp = 4.187 kJ/kgK

$\Delta T$ = (10-0) = 10 degrees C

q = 30 X 4.187 X 10 = 1.2 MW

For Ice:

m = 30 kg/s

334 kJ/kgK = latent heat of fusion (water)

q = 30 X 334 = 10.0 MW

Total heat transferred to air = 11.2 MW

For Air: $q = mC_p\Delta T$

m = q / (Cp X $\Delta$T)

m = 11.2 / (1.005 X 25) = 446 kg/s or 371 $m^3/s$ @ 1.20 $kg/m^3$

With the availability of an ice stope, the mine is capable of heating approximately 371 m3/s from -20 to +5 degrees C for the cost of pumping 30 liters per second of wastewater! Assuming a calorific value of 25.3 MJ/liter for Propane, this amount of heating is equivalent to almost 1,600 liters per hour and almost $600,000 per month at an estimated cost of around 0.5 $/liter of Propane.

## Case Study: Zinkgruvan Mine


The Zinkgruvan Mine is a base-metal mine located in southern-central Sweden near the town of Åmmeberg on the northern end of Lake Norra Vättern. The mine has been in operation since the 18th Century, and now extends to more than 1,200m below the surface.

In 2013, the Mine installed an innovative waste-heat recovery system at the Kristina Intake and Exhaust Shaft locations. Warm air from the mine exhaust passes through an indirect, plate-type heat exchanger before exhausting the system. Fresh air is pulled through the heat-exchanger prior to passing through the intake fan and being blown into the mine. Air through the intake and exhaust shafts are relatively balanced, at approximately 320,000 cfm each.

In order to prevent the mine services from freezing, it is necessary to heat the mine intake air to a minimum of 37 degrees Fahrenheit. Air exits the mine at a relatively consistent 52 degrees Fahrenheit. The heat-exchanger is capable of meeting the minimum intake air temperature requirements during environmental conditions that range down to 16 degrees Fahrenheit, below which temperature the back-up oil-fired burners will provide additional air heating.

The waste-heat recovery system installed at Zinkgruvan has reduced the fuel oil consumption of the Kristina Intake Air Heaters by 97% (300,000 liters of fuel oil per year), providing a payback of the initial investment in less than two years. The heat exchanger has now been in operation for more than three years without any significant operational problems and is considered a model for how the mine can reduce operating costs and improve efficiency at several other ventilation installations. This method of waste-heat recovery can be implemented at any operating mine where the distance between the intake and exhaust shafts is not too great, or where other sources of waste-heat are available.

# Diesel Particulate Matter

Jump to Today | Edit

The AEOLUS Project

**Health Effects, Measurement and Control of Diesel Emissions in Underground Mines**

Diesel-powered equipment is ubiquitous in today's mechanized underground mines.  While largely responsible for the efficiency and productivity of modern mines, diesel equipment also produces significant hazards to human health, especially in sub-surface environments.  These hazards present a challenge to mine ventilation engineers and technicians tasked with protecting the health and safety of the workforce. This course provides an overview of the desultory health effects, measurement techniques and control of diesel emissions in underground mines and facilities.

**Learning Objectives**

1. **Identify emissions control technologies.**
2. **Identify health risks associated with diesel emissions exposures.**
3. **Explain the role of environmental sampling in diesel emissions control.**
4. **Identify four types of contaminants produced by diesel equipment underground.**
5. **Explain the ALARA principle as it relates to diesel emissions control.**
6. **Demonstrate knowledge of engineering controls for reducing DPM emissions.**

Get Started

Go to the Course Index

About the Aeolus Project

## Course Summary:

| Date | Details |
| --- | --- |
| | Diesel Emissions Wrap-up (https://canvas.instructure.com/courses/1049433/assignments/5490449) |

# 1.0 History of Diesel Engines in Underground Mines

The use of diesel-powered equipment in underground mines today is nearly ubiquitous. Diesel engines power mine equipment from the smallest horsepower generators and pumps, to 50-ton haul trucks and everything in between. There are many reasons for this almost universal adoption, including the flexibility, power and availability of the diesel powerplants which can be applied to the full spectrum of equipment required in modern mines.

If current trends in the mining industry continue, both the number and the size of underground diesel equipment will continue to increase in the future, making the mitigation of diesel emissions and associated contaminants an ever-increasing concern for ventilation engineers. There is also some evidence to suggest that the technological complexity of newly-engineered diesel powerplants and their associated emissions controls (i.e., Tier IV, Stage IV Euro) are more sensitive to ambient heat and dust, leading some experts to believe that the ambient intake conditions may become a critical parameter for ventilation system design in the near future.

# 1.1 Development of the Diesel Engine and Diesel Engine Technology

Rudolf Diesel was awarded a patent for an internal combustion engine based on the Carnot cycle in 1892. A working prototype of this invention followed in 1897. The
“diesel” engine would go on to become a transformative machine in the course of human history- ushering in a wave of industrialization that has yet to see its utility diminished or replaced by any other machine. Today, the diesel engine is a mainstay of the mining industry, powering almost every piece of moving equipment in the mine, from the smallest portable generators and compressors to the largest of the haulage fleet, the diesel engine is everywhere. Understanding how the diesel engine has changed and improved in the decades since its humble beginning is necessary in order to fully comprehend this current mainstay of mine power production.

Figure 1.1 depicts two versions of the diesel engine developed approximately 90 years apart.

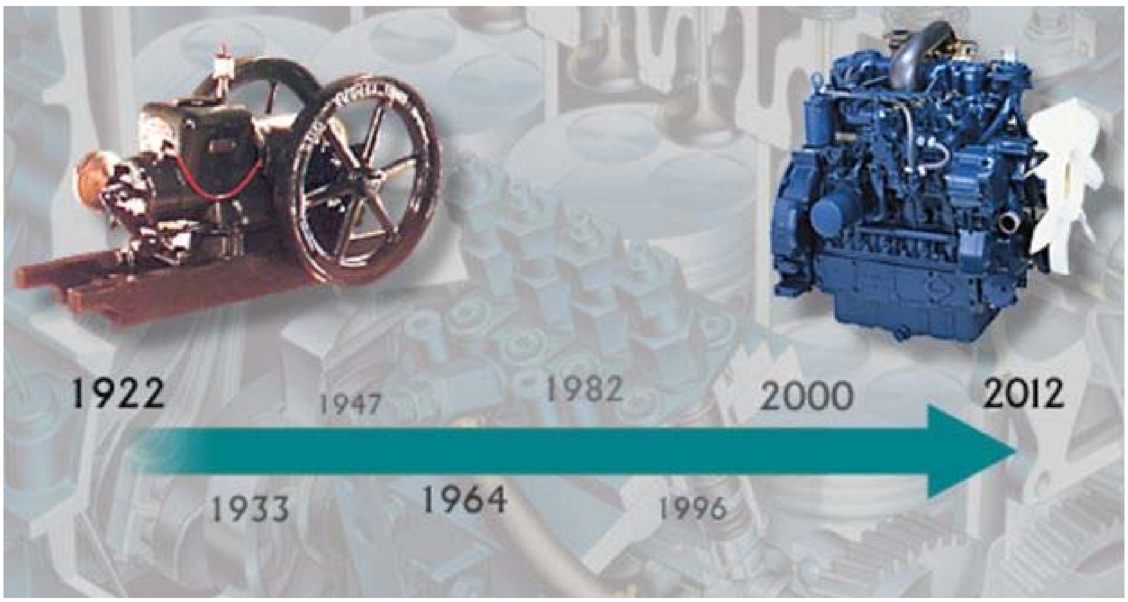


**Figure 1.1: Diesel Engines circa 1922 and 2012 (Kubota Corporation).**

The first diesel engine, based on the Carnot cycle represents the maximum possible efficiency for any internal combustion engine (note that the efficiency of an actual diesel engine is somewhat less than the theoretical maximum defined by Carnot). When compared to Otto cycle engines (i.e., gasoline-powered) the diesel engine provides numerous advantages in industrial applications where it can be adapted to a variety of purposes.

The diesel cycle consists of four “strokes” that define the activities of the engine during each step; induction, compression, expansion or “power” and exhaust that occur as a piston moves up and down in a cylinder. The piston is attached to a rotating shaft which serves as a convenient point for power take-off.

Figure 1.2. depicts a pV diagram for the Diesel Cycle.

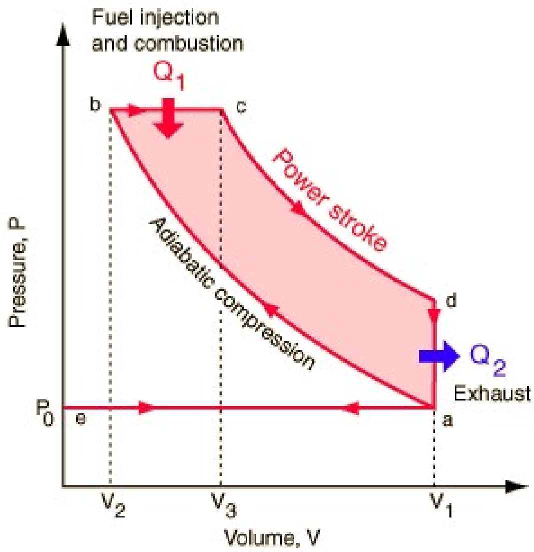


**Figure 1.2: The Diesel Cycle (Georgia State University).**

A cross-sectional schematic of typical diesel cylinder and piston is shown on Figure 1.3.

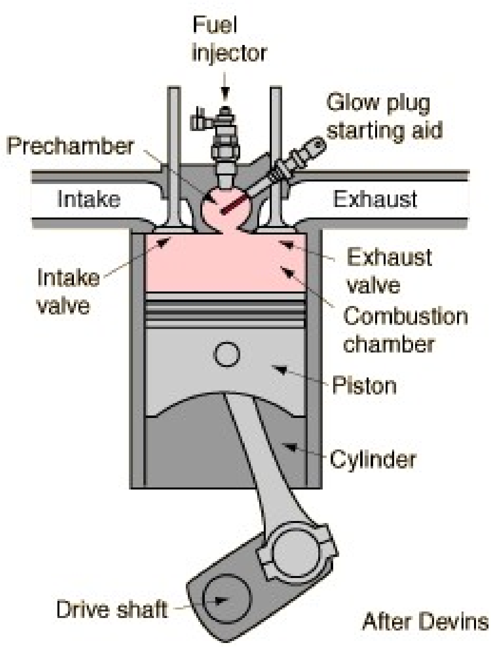


**Figure 1.3: Typical Diesel Engine Cylinder/Piston (Georgia State University).**

The induction stroke starts at a theoretical point at the topmost extent of travel sometimes refered to as Top Dead Center (TDC). The piston travels down through the cylinder as the intake valve opens and allows ambient air to enter the cylinder.

Figure 1.4 illustrates the induction stroke of a diesel engine.

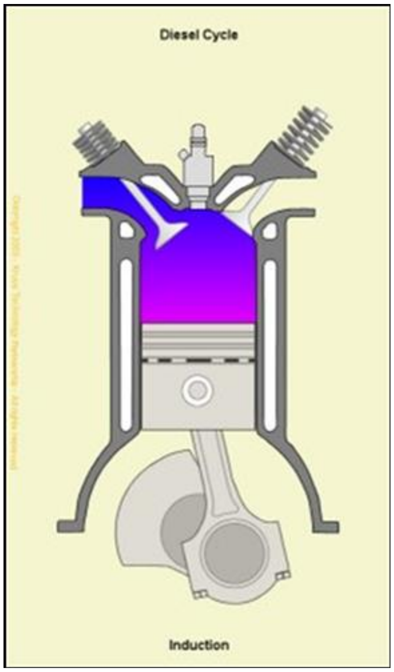


**Figure 1.4: The Induction Stroke (Kruse Technology**).

Immediately following the induction stroke, the piston travels back up the cylinder from Bottom Dead Center (BDC) with both valves closed- resulting in the adiabatic compression of the air contained within. Fuel is injected into the cylinder at the very end of the stroke (at TDC) which is immediately ignited.

The compression stroke of a diesel engine is illustrated on Figure 1.5.

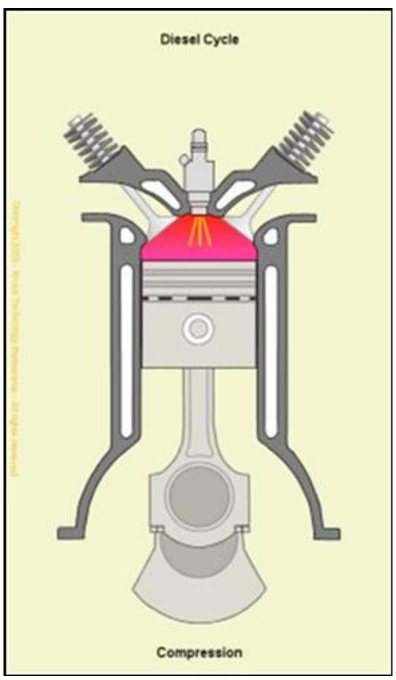


**Figure 1.5: The Compression Stroke (Kruse Technology).**

In the power stroke, the ignition of the fuel and air causes the rapid adiabatic expansion of the mixture as the piston travels from TDC to BDC.

The power stroke of a diesel engine is shown on Figure 1.6.

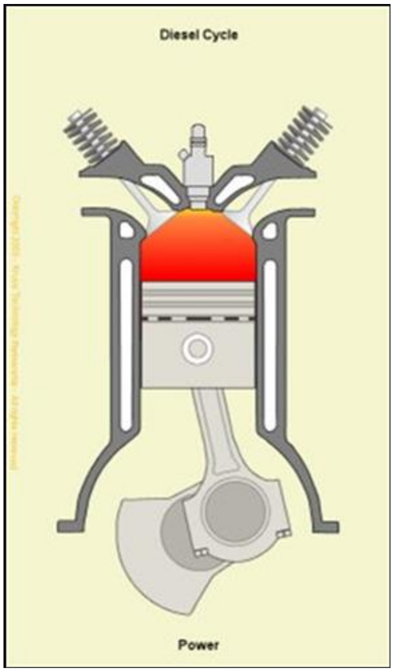


**Figure 1.6: The Power Stroke (Kruse Technology).**

As the piston begins to travel back from BDC to TDC the exhaust valve at the top of the cylinder, pushing the products of combustion out on the way up.

The exhaust stroke for a diesel engine is depicted on Figure 1.7.

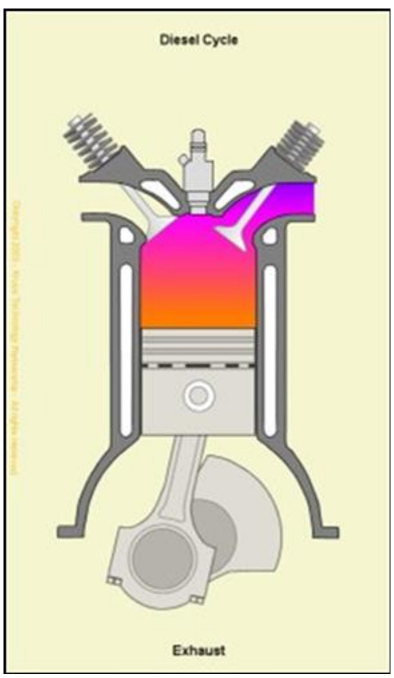


**Figure 1.7: Exhaust Stroke (Kruse Technology).**

Once the exhaust stroke has been completed the entire cycle is repeated. The simplicity of Diesel's design, coupled with the ease with which his engine could be built and operated led to their rapid acceptance and use in a variety of heavy industries.

Over the next seventy-five years, the only real changes made to the original design were to make incremental improvements to the various components, or to scale them up to in ever-increasing sizes and power outputs. It was during this time that the original turbocharger was invented (1906), followed by the first common rail induction system by Clessie Cummins in 1946 (Cummins, 1998).

The next monumental advancement in the technology of the diesel engine came about in the 1970s, when economic pressures brought on by the oil embargo of 1973 pushed engine manufacturers to increase efficiencies. It was then, some seventy years after its invention, that the turbocharger began its reign of popularity with engine manufacturers as well as consumers.

A turbocharger, or “turbo” utilizes exhaust air to spin a turbine that compresses the intake air prior to its induction into the cylinder. The increased air density of the intake can allow for higher fuel dosing and greater power output.

The benefits of fitting a diesel engine with a turbocharger include:

- Increased power from a similarly sized engine, or…
- Equal power output from a smaller engine.
- Increased torque at lower engine speeds.
- Quieter engines
- Better fuel economy
- Less noxious exhaust

Following the success and widespread acceptance of the turbocharger, and the rapid expansion of the new, more efficient diesel vehicles world-wide, the traditional means of controlling the timing of the engine were unable to support the demands of the newer, more efficient engines (Stinnette, 2013). The German automotive company Bosch invented the first module that allowed electronic engine control (EEC) for diesel passenger cars. This was followed in 1989 with an EEC module for commercial vehicles. This EEC system provided much greater control over a variety of engine functions, and resulted in another significant improvement to the efficiency of diesel engines that employed the technology (Bosch, 2006).

Following the widespread adoption of both turbochargers and EEC modules by commercial and industrial engine manufacturers and users, the industry would be faced with their greatest challenge yet.

In 2004, the U.S. EPA and the EU enacted legislation that drastically limited the permissible gaseous and particulate emissions from diesel engines as part of a global initiative to reduce greenhouse gases and combat climate change. In order to meet the requirements of the EPA’s “Tiered” emissions reduction plan, almost every component of the diesel engine would have to be improved and remade.

The EPA and EU-mandated reductions to the engine-out emissions of diesel equipment were applied evenly to all manufacturers, and unlike mining regulations, gave no consideration to feasibility or cost. In order to meet these stringent new requirements, it was necessary to completely re-engineer almost every component of the engines and several other related systems (e.g., intake, exhaust).

As the first of the EPA Tiered regulations were implemented, engine manufacturers began to refine and optimize the performance of their existing engines. Since the regulations were based solely on the engine-output, manufacturers were free to experiment with the various technologies, and mix and match control devices as they wanted in their quest to improve the emissions profile of their products.

Figure 1.8 illustrates the relationship between engine-out $NO_x$ and DPM emissions.

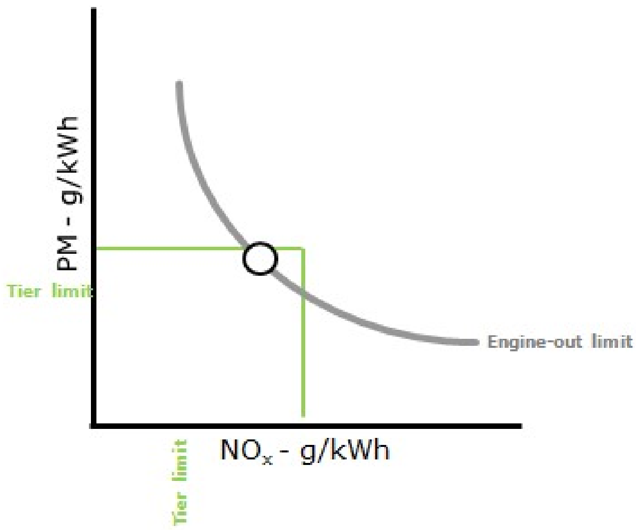


**Figure 1.8: DPM versus $NO_x$ production in diesel engines (John Deere).**

As illustrated in the above figure, the DPM and NOx produced by a diesel engine are inversely proportional. As the ignition timing is advanced or retarded, the total emissions move up and down the curve. This relationship partially explains how some of the interim-compliant engines experienced increases in $NO_x$ production, as engine timing was experimented with via the EECs.

An additional innovation that was implemented at this time was the high-pressure common rail (HPCR) injection system, which utilizes a specialized, high-pressure fuel pump to generate fuel pressures of 20,000 to 30,000 psi in an injection system that is shared among each cylinder. In conjunction with the EEC, the HPCR system allows incredible flexibility in the control of each individual cylinder.

A schematic showing the components of a diesel-engine HPCR system is given on Figure 1.9.

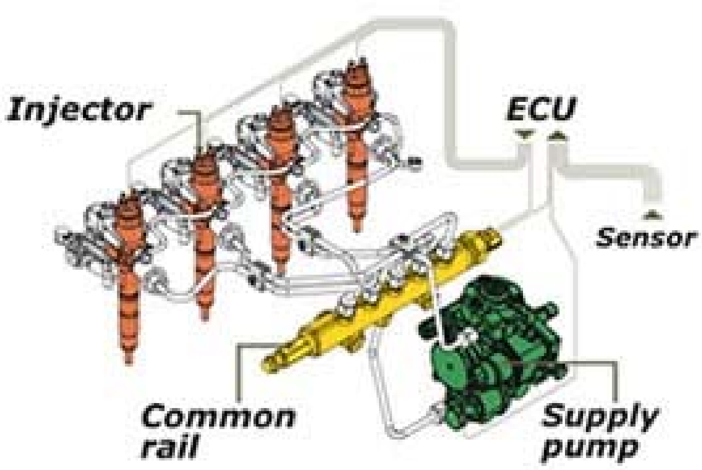


**Figure 1.9: Diesel-engine HPCR system (Kubota).**

Charge air coolers, or CACs, have a dual purpose when fitted to diesel engine. The CAC cools the charge air prior to entering the cylinder for combustion, with the denser air responsible both for greater combustion efficiency and lower $NO_x$ emissions (due to the reduced temperature of combustion).

Figure 1.10 depicts a diesel engine schematic including a charge air cooler.

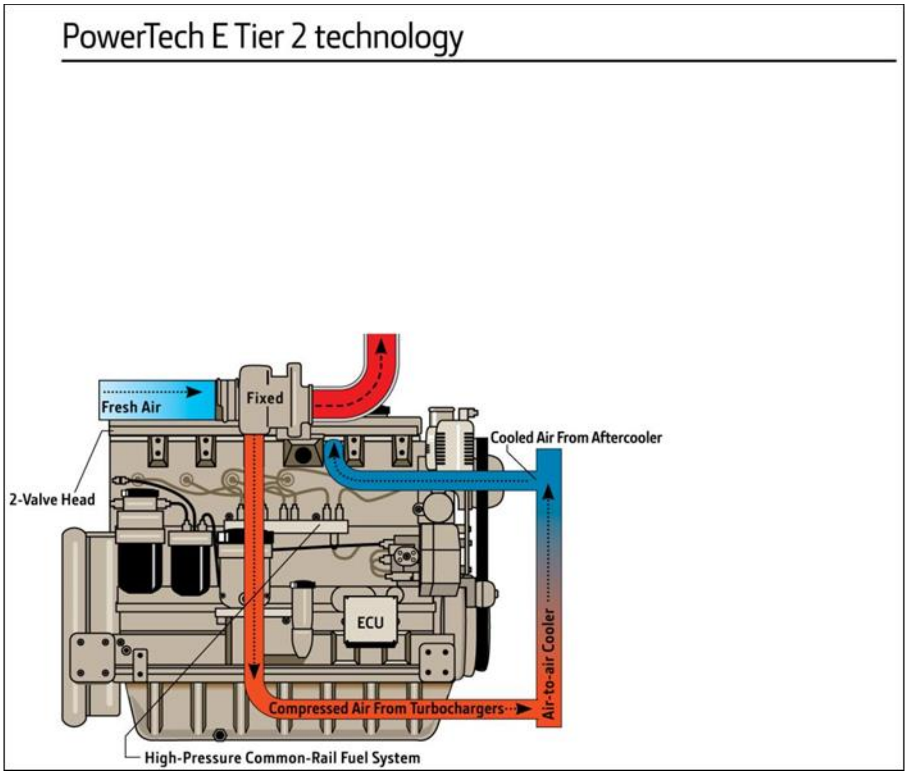


**Figure 1.10: Diesel engine schematic with CAC included (John Deere).**

As ambient conditions around the equipment change, the condition of air being supplied to the engine also changes. In order to provide a more consistent air temperature and density for the cylinders (again, to maximize efficiency), variable-geometry turbochargers VGTs were introduced.

Another method for reducing the temperature at which combustion occurs in the cylinder is to recirculate some of the exhaust gas back into the charge air (after cooling). The resulting intake air has less $O_2$ (replaced with $CO_2$ from the cylinder exhaust), which results in lower NOx production in engines equipped with this technology of exhaust gas recirculation, or EGR.

Figure 1.11 shows the addition of a variable-geometry turbocharger and exhaust gas recirculation to our diesel engine schematic.

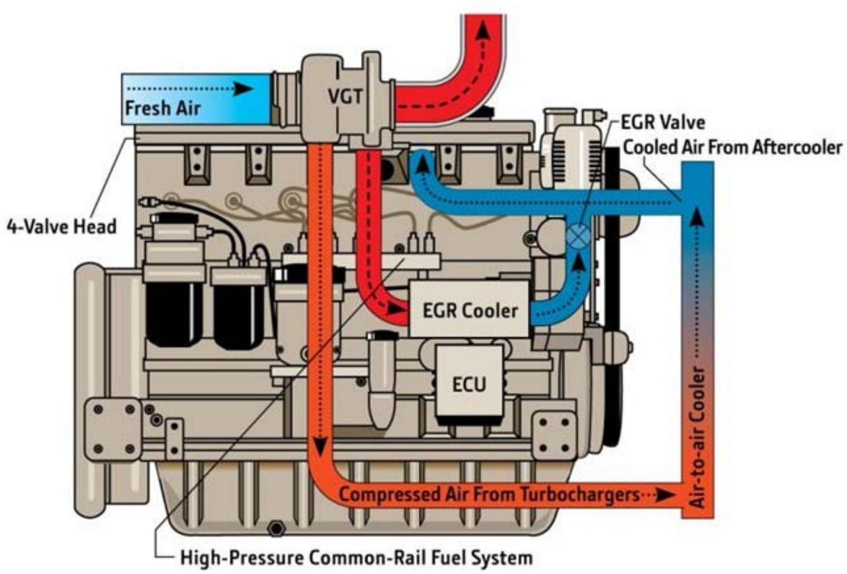


**Figure 1.11: Diesel engine schematic showing VGT and EGR (John Deere).**

High efficiency diesel particulate filters (DPFs) have been shown to remove as much as 99% of the engine-out DPM by weight. As such, they are often included as part of the diesel engine exhaust system with an in-line diesel oxidation catalyst (DOC) that aids in the conversion of CO present in the tailpipe to less toxic $CO_2$.

In order to meet compliance with EPA Tier IV regulations, any exhaust after-treatment that is included in the approved engine package must be submitted individually for certification in conjunction with that engine and maintained appropriately (may not be removed or altered).

The diesel engine schematic with an in-line DOC and DPF attached is shown on Figure 1.12.

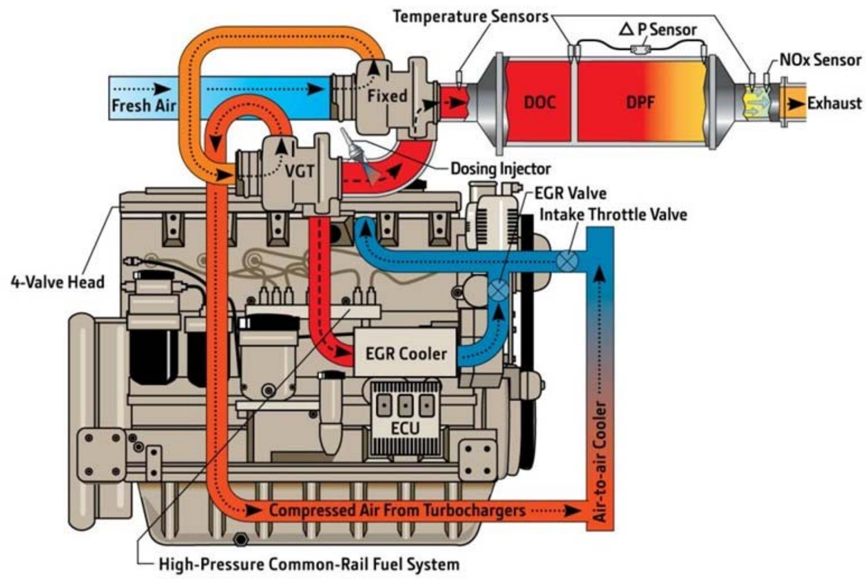


**Figure 1.12: Diesel engine schematic with in-line DOC and DPF (John Deere).**

An additional exhaust after-treatment technology that may be implemented either separately or in conjunction with the in-line DOC and DPF is called selective catalytic reduction or SCR. When installed in the exhaust system of a diesel engine, the SCR hydrolyzes $NO_2$ in the engine exhaust into ammonia and water vapor by injecting a urea solution directly into the exhaust.

The SCR system attached to the diesel engine exhaust is shown on Figure 1.13.

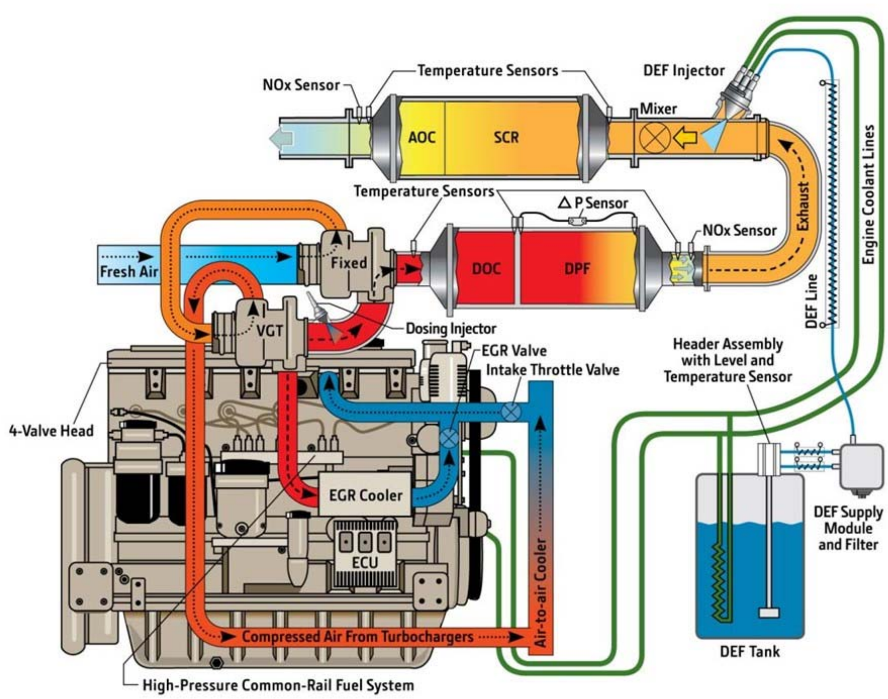


**Figure 1.13: Prototypical Diesel Engine with SCR System (John Deere).**

When considering the scope of the final Tier IV regulations, as well as the difficulties involved in reducing both the DPM and NOx simultaneously, it makes sense to employ multiple, targeted control strategies in any one engine package submitted for approval.

Figure 1.14 revisits the DPM versus $NO_x$ curve from above, but with the effects of the various emissions-reduction technologies called out directly.

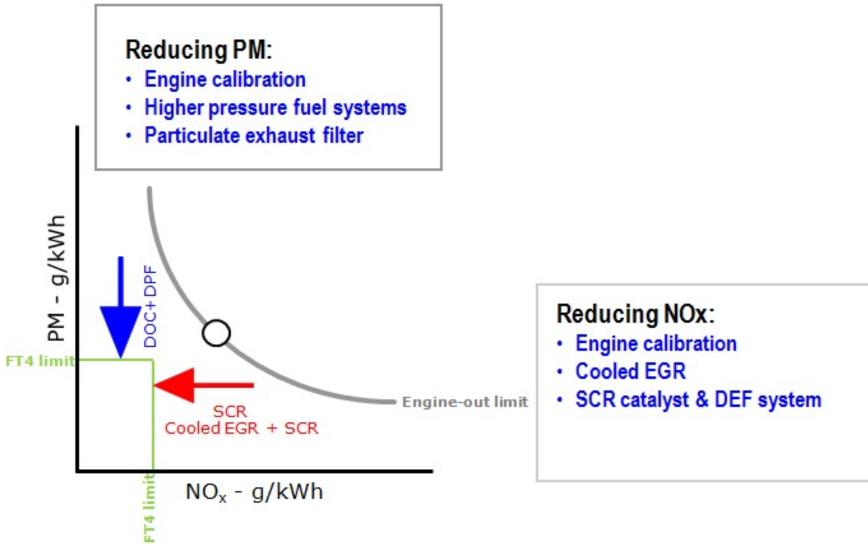


**Figure 1.14: Effect of various emissions-control technologies on DPM and $NO_x$ (John Deere).**

The result of all this variation in technology and results is that the Tier IV compliant engines from various manufacturers are all equally varied- with some manufacturers utilizing different technology within different product ranges and sizes. Thus, for those required to maintain these engines in the mining environment, the task has become exponentially more challenging, and usually, more expensive.

# 1.2 Use of Diesel-Powered Equipment in Underground Mines

The history of diesel engine use in underground mines extends for almost 90 years.  Starting with the first diesel engine used underground in Germany in 1927, diesel equipment began to be slowly introduced into underground mines around the world.

Table 1.1 gives an early timeline of diesel engine use in underground mines.

**Table 1.1: Timeline of diesel use in underground mines (Kenzy and Ramani, 1980).**

| | |
|---|---|
| 1882 | Diesel Engine Invented |
| 1886 | First Gasoline Locomotive in an Underground Mine (Germany) |
| 1897 | Diesel Engine Reduced to a Practical Size |
| 1906 | First Gasoline Engine in a U.S. Mine |
| 1915 | Most States in the U.S. Outlaw Gasoline Engines Underground |
| 1927 | First Diesel Engine in an Underground Mine (Germany) |
| 1934 | Diesels in Belgian, British, and French Underground Coal Mines |
| 1939 | First Diesel Engine in a U.S. Underground Mine (Pennsylvania) |
| 1946 | First Diesel Engine in a U.S. Underground Coal Mine |
| 1950 | Development of the First Diesel-powered LHD |

As technology improved, and mine output soared, diesel engines became widely accepted in the mining industry in the 1960s. From that time on, the use of diesel-powered equipment has grown, with more and larger pieces of equipment year on year.

There are many reasons for the popularity and widespread use of diesel-powered equipment underground.  Diesel engines are more efficient, more powerful and safer than their gasoline equivalents.  Unlike electric equipment, they are not limited in travel by a power cable and have no need to spend significant down-time recharging or swapping batteries.

At present, specialized equipment powered by diesel engines exist for most tasks performed during the underground mining cycle; including diesel loaders, haul trucks, bolters, shotcrete sprayers, ANFO loaders, scissor lifts, secondary blast rigs, jammers, forklifts, graders and personnel carriers to name a few.

# 1.3 Regulation of Diesel Equipment/Emissions in Underground Mines

The testing of diesel engine emissions was conducted by the US Bureau Mines in the 1950s under 30 CFR part 32 Schedule 24. This statute was later superseded by Part 7 to allow for third-party testing of diesel engines (Haney, 2012).

Gaseous tailpipe emissions of diesel equipment were first regulated in the U.S. in 1973. The first time particulate emissions were specifically included in U.S. regulations came in 1988, when the MSHA recommended a three-part approach to the control of diesel emissions underground following decisions by the International Agency for Research on Cancer (IARC) and the National Institute for Occupational Safety and Health (NIOSH), that diesel emissions constituted a “probable” carcinogen. These regulations set Threshold Limit Values (TLVs) for Carbon Monoxide (CO), Carbon Dioxide ($CO_2$), Nitric Oxide (NO) and Nitrogen Dioxide ($NO_2$).

At this time, there was still no accurate method for determining diesel particulate exposure levels, and no established criteria for its regulation. It was not until ten years later, that a team of NIOSH scientists led by Dr. Eileen Birch developed a way to accurately measure ambient levels of diesel particulate matter (DPM), the so-called NIOSH 5040 Method (NIOSH, 1998).

In the intervening period, DPM levels were determined gravimetrically by weighing sample collection filters before and after being placed in a 400 C oven for two hours. This technique, known as the Respirable Combustible Dust (RCD) method is highly susceptible to corruption by other combustible dusts, aerosol oils and other combustible fumes (e.g., cigarette smoke, welding fumes, etc.). Even then, its precision only allows determination of the RCD to approximately +/- 40 µg Carbon (equivalent).

The NIOSH 5040 Method involves the use of a thermal-optical analyzer to determine the amount of Organic and Elemental Carbon (C) present on a known size punch of a quartz-fiber filter. The carbon content is measured in micrograms (µg) per square centimeter of filter, and when multiplied by the total filter size and divided by the total volume of air passed over the filter during the sample period, an ambient exposure level of Carbon (used here as a surrogate for DPM) is obtained in units of µg Carbon per cubic meter of air.

This method of sampling and calculation (which will be further covered in subsequent sections in greater detail) has since become the standard for measuring ambient DPM levels around the world owing to the precise nature of the results.

A further ten years later in 2001 the MSHA began to phase in a DPM TLV in underground metal/non-metal mines in the U.S. The TLV for ambient DPM exposure would fall from 400 $µg/m^3$ to 160 $µg/m^3$ over a period of six years. The method for determining exposure levels for enforcement purposes was specified as the NIOSH 5040 Method. Further provisions to the regulations were added that governed the frequency of DPM sampling by mines and for annual safety training of underground personnel regarding the health risks associated with diesel emissions.

It was also at this time that the MSHA enacted regulations governing the production of DPM in underground coal mines. Although the precision-jeweled impactor cassette specified in the NIOSH 5040 Method was specifically designed for the collection samples of DPM in coal mines (allowing only the collection of sub-micron particles to pass) the potential for contamination from coal dust led to a different set of regulation for coal mines. Those regulations focus on airflow rates for approved engines (based on laboratory benchmark tests) and applied exhaust after-treatment devices (along with similar provisions for employee training).

Other countries around the world soon followed suit, and enacted, or altered their mining regulations to include similar restrictions to the amount of DPM exposure that was acceptable for underground mine workers.

In Germany and Switzerland the maximum allowable DPM exposure is 100 $µg/m^3$ Total Carbon (TC) in all underground non-coal mines and tunnels. In Australia, the national institute of occupational hygienists (AIOH) stipulates that mine workers should not be exposed to more than 100 $µg/m^3$ of Elemental Carbon (EC) (AIOH, 2007).

Notable exceptions to this trend in the Western World include the provincial regulatory bodies in Canada, who still allow DPM exposure levels to be determined by the RCD method at up to 1,200 $µg/m^3$ with the exception of Ontario who sets their limit at TC or 1.3 X EC < 400 $µg/m^3$ (Ontario MOL, 2011).

Current TLVs for the gaseous components of diesel emissions in underground mines mirror those recommended by the ACGIH;

presently 25 ppm CO, 5000 ppm $CO_2$, 25 ppm NO and 3 ppm $NO_2$. Most other developed nations have adopted similar regulations and limits. In Germany, a leader in this field, the TLV for $NO_2$ is expected to be further reduced to <1 ppm, problematic since that is a level that is not currently possible to measure (Dahmann, 2010).

# 1.4 Regulation of Non-road Diesel Engines Affecting Mine Equipment Fleets

It is arguable that the most impactful regulation of diesel mining equipment world-wide comes not from any mining regulatory body, and was not specifically targeted towards mines, or mine equipment.  Nonetheless, the regulation of non-road diesel equipment emissions that were first implemented by the U.S. EPA (and mirrored by the European Equivalent EURO regulations) in 2004 have had led to the most significant changes to diesel engine technology and the greatest reductions in engine-out emissions in the history of the technology.

The U.S. EPA first began regulating the emissions of non-road diesel engines in 1994.  In 1996, they issued a “Satement of Principles” that was signed by the EPA, and 13 of the top diesel engine manufacturers, among others (Dieselnet, 2012).  The EPA outlined an extensive program designed drastically reduce engine-out DPM and gaseous emissions, while also making other stipulations on the fuel quality used by these engines.  The reductions were to be implemented in a phased approach, with different guiding metrics to affect different engine types and outputs, that hereafter became known as the “Tier I through Tier IV” (or EURO Stage I – Stage IV) regulations.

Tier I standards were begun in 1996, with Tier II beginning in 2001.  Tier III standards covered the time period between 2006 and 2008, before the final Tier IV regulations began (first with Tier IV Interim) and finally culminated with the final rules to be enforced in 2015 (and after).

Tier IV Final regulations mandate the reduction of $NO_x$ and DPM by approximately 90% over pre-tier levels.

During this time period, the EPA also introduced new guidelines limiting the sulfur content of diesel fuel (associated with organic carbon PM emissions) for non-road applications.  Beginning in 2007, the sulfur content of non-road diesel was limited to 500 ppm, and by 2010 this level was further reduced to 15 ppm.

A further requirement of the EPAs regulations governs the approved method(s) for testing the compliance of non-road diesel engines.  In order to comply with the Tier I – IV standards, engines must be tested with the ISO 8178 C1, also known as the 8-mode, steady-state protocol and the Non-road Transient Cycle Test.

Figure 1.15 shows how the EPA non-road emissions regulations were implemented (Tier I – IV).

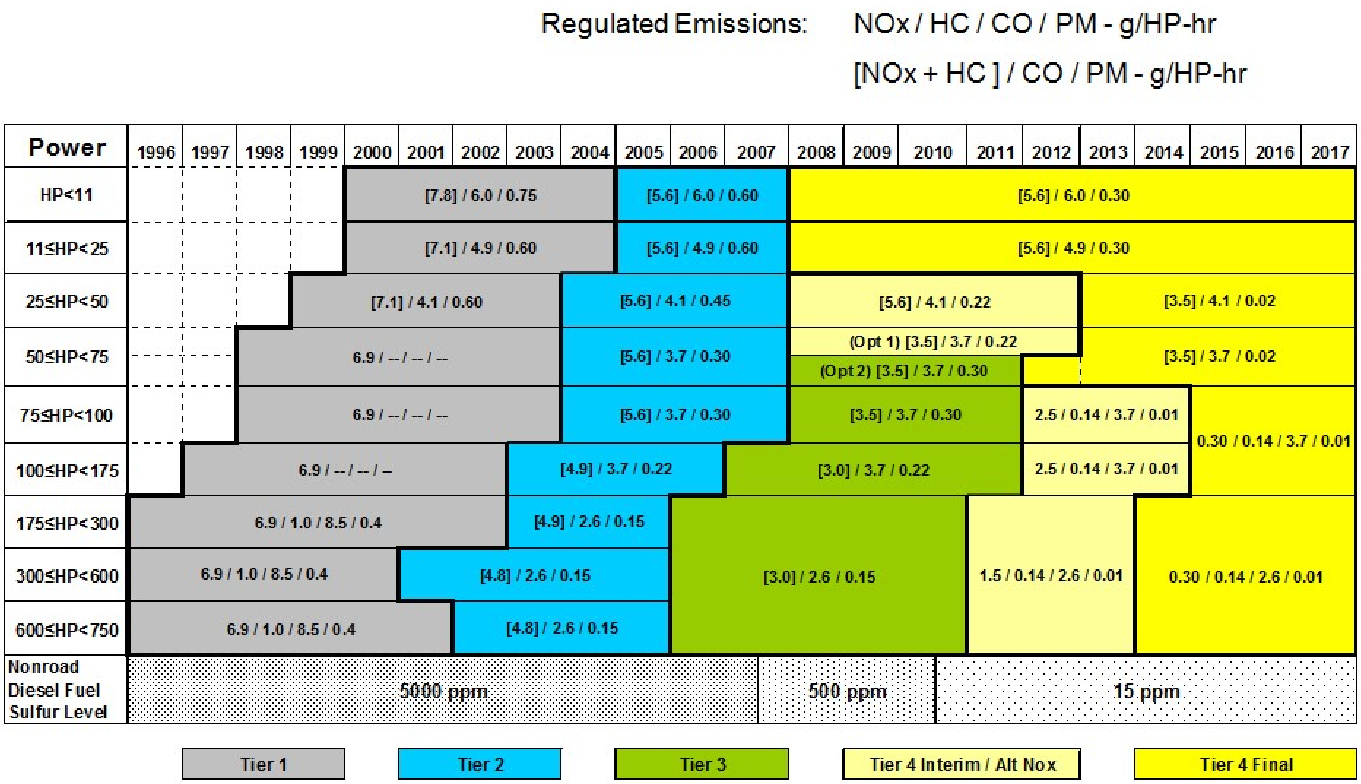


**Figure 1.15 is a graphic representation of the scale of the DPM and $NO_x$ reductions that were mandated.**

Figure 1.16 shows EPA Requirements during the Transitional Period from Tier I – IV.

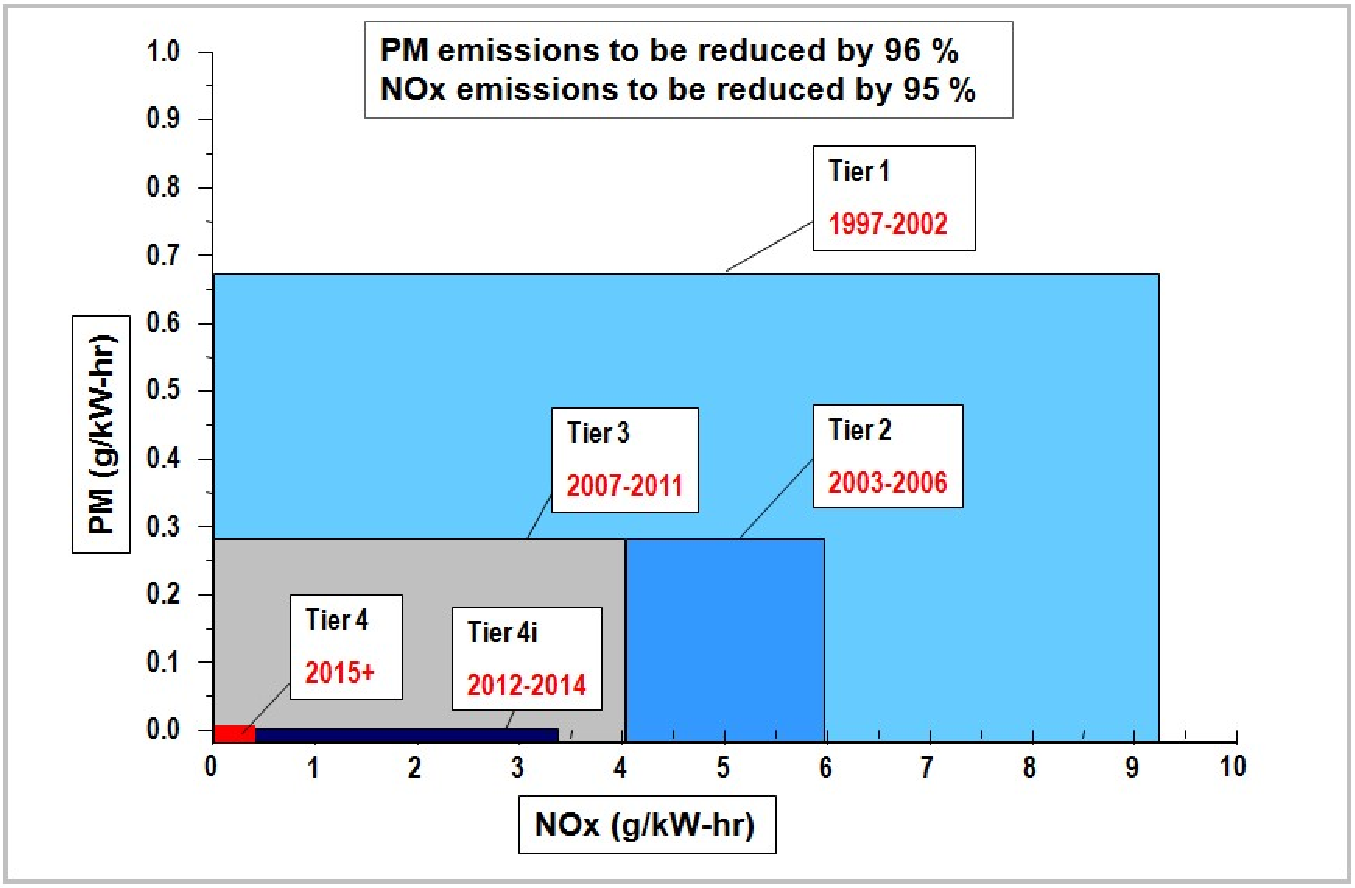


**Figure 1.16: Required emissions reductions from non-tier to Tier IV levels (Deutz).**

Other nations around the world have adopted their own strategies for non-road diesel emissions regulations and reductions. Many have adopted the U.S. EPA (or similar) standards, while some have no regulations at all. A thorough knowledge of the local regulations is imperative for establishing the design criteria for a ventilation system design. In many cases, large, multinational corporations will voluntarily adopt more stringent standards for design that what they are legally obligated to meet.

Figure 1.17 shows some of the non-road diesel emissions regulations for varius countries and regions.

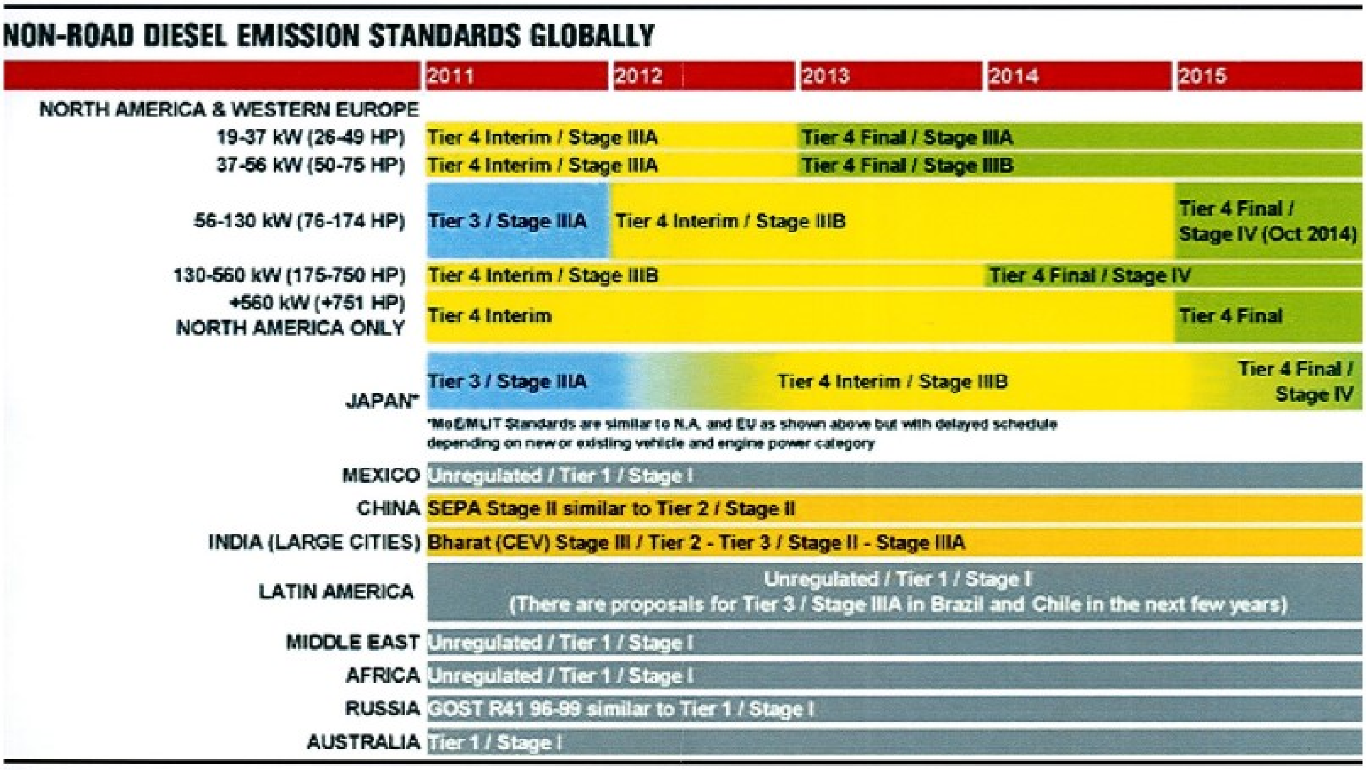


**Figure 1.17: Non-road diesel emissions regulations for various regions (Kubota Corporation).**

# 2.0 Health Effects of Diesel Emissions

Diesel (exhaust) emissions constitute a mixture of gaseous and particulate components that each represent a particular as well as a combined health risk to those exposed. In addition to the tailpipe emissions, diesel equipment that is operated in subterranean environments also produces significant heat and dust, which if not mitigated will also pose a risk to the underground workforce.

All four of these types of contaminants must be addressed by the ventilation system of any underground mine or facility where present, with each having unique risks and mitigation strategies.

The output of harmful gases and particulates from diesel engines can vary greatly, even among engines of similar size, based on the manufacturer, age, and maintenance of a particular engine.

Dust generated during the operation of the equipment (e.g., loading, trammming, dumping, etc.) will also vary in the threat it poses, from nuisance dusts (e.g., limestone, potash, etc.) to those which represent an extreme and acute danger (e.g., asbestos, high silica content, radon daughters, etc.).

The impact on the environmental quality from heat generated by diesel equipment can in some cases have a positive impact (e.g., reducing the amount of bulk air heating required in cold-weather climates) but in others can represent the single greatest demand on the ventilation system and the limiting factor in mine development, particularly in deep mines.

An understanding of how these contaminants are produced as well as their relative risk to health and safety is vital in developing a plan for control and reduction of these potentially harmful by-products of diesel equipment use underground.

# 2.1 Gaseous Emissions

The gaseous emissions from diesel engines are primarily made up of carbon monoxide, carbon dioxide, nitric oxide, nitrogen dioxide and water vapor. While other potentially toxic gases may be present, they are generally not found in concentrations significant enough to cause concern for ventilation system designers.

Carbon Monoxide (CO) is produced by incomplete combustion in diesel engines. It is odorless, colorless and inflammable. CO is also explosive in air in the range of 12. To 74%. It's only slightly less dense than ambient air, mixes readily and will not collect along floors or low-lying areas. The physiological effects of CO poisoning (headache, nausea, dizziness and fatigue) are similar to those of many common maladies, leading many exposed to misdiagnose or even fail to treat their condition(s) altogether. Once inhaled into the lungs, CO attaches itself readily to red blood cells, where it then prevents the adsorption and transmission of oxygen throughout the body. Cells are slowly deprived of oxygen, which results in the symptoms described above. Death is possible in ambient concentrations of CO as low as 400 ppm, and will occur much more rapidly at higher concentrations (less than one hour at 1,500 ppm). Those exposed to higher concentrations of CO will often fatigue quickly, fall asleep, and expire without ever regaining consciousness.

Carbon Dioxide ($CO_2$) like CO is odorless, colorless, but is not flammable or explosive. It is significantly denser than air, and will readily collect along the floors and low-lying areas in mines where it is present in significant concentrations. $CO_2$ starts to affect the body in concentrations as low as 2%, and initially acts as a stimulant on the respiratory system, increasing the breathing rate. Breathing rate will continue to increase as $CO_2$ concentrations increase, with dizziness, nausea and a loss of consciousness occurring between 10 and 20%. Death can occur quickly in concentrations greater than 20%.

Oxides of Nitrogen, and $NO_2$ in particular are highly toxic gases that are present in diesel engine exhaust. $NO_2$ has a reddish brown color in high concentrations, and has a slight "acidic" scent. $NO_2$ reacts in the human respiratory system to produce nitric acid that results in pulmonary edema. $NO_2$ can be toxic to humans in concentrations less than 2 ppm. Irritation of the nose and throat occurs in concentrations of 10 ppm, followed by tightness in the chest at approximately 80 ppm. Death from pulmonary edema is possible in concentrations as low as 90 ppm in just half an hour of exposure.

# 2.2 Particulate Emissions

The particulate emissions from diesel engines, commonly referred to as diesel particulate matter (DPM) constitute a two-pronged threat the health of humans. Firstly, the particles that constitute DPM are generally less than one micron in diameter; second, these tiny particles adsorb a variety of toxic chemicals (e.g., aromatic hydrocarbons, aldehydes, etc.). In combination, DPM carries these harmful chemicals deep into the lungs and respiratory system, where they can be transported in to the body.

Figure 2.1 shows a graphical representation of DPM as it occurs in diesel exhaust emissions.

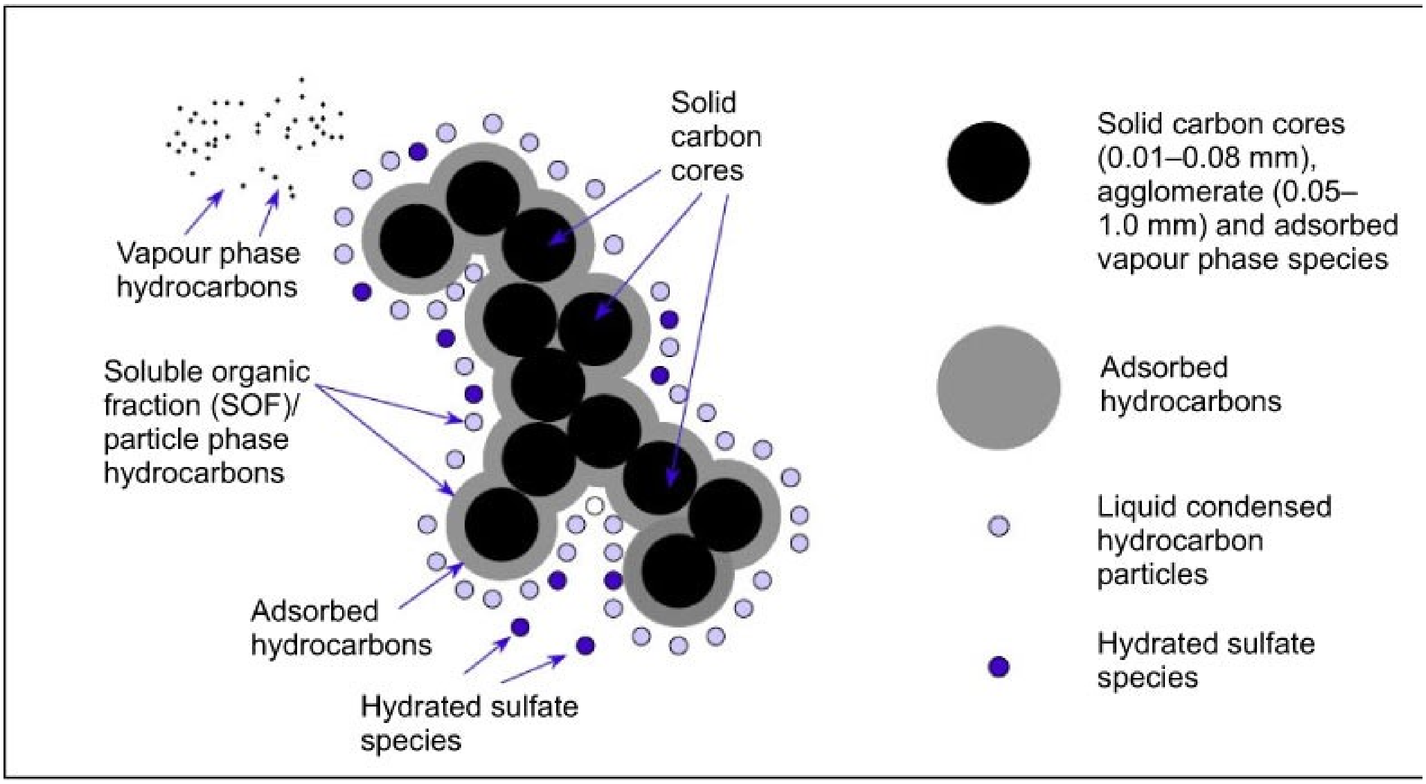


**Figure 2.1: Aerosol diesel particulate matter (Twigg and Phillips, 2009).**

In 2012, the results of two, independent health studies were concluded; one by the National Institute for Occupational Health and Safety (NIOSH) and one by the International Agency for Research on Cancer (IARC), which both revealed that DPM represented a more dangerous health threat than was previously acknowledged.

IARC, a division of the World Health Organization (WHO), is an inter-disciplinary organization that coordinates cancer research and focuses on identifying the sources of cancer around the world. Indirectly, through the WHO, IARC also helps to guide health and safety regulations and policy.

The results of the landmark IARC study on diesel emissions released in June 2012 recommended that diesel engine exhaust be classified as a human carcinogen based on the evidence collected by the study. This represented a significant change from the agency's previous classification of diesel exhaust as a "probable carcinogen", and parked a global change in the way that diesel emissions are classified and regulated thanks to the agency's influence on policymakers. This influence is especially powerful in the developing world, where governments often lack internal organizations devoted to public health research. Even in the developed world, organizations like the NIOSH and U.S. EPA generally follow the recommendations of IARC with regard to the classification of substances that are classified as carcinogens or potential carcinogens.

Notwithstanding the excellent work that has been done to date by NIOSH and IARC, the physiological threat from diesel engine emissions is still only partially understood, and consequently the subject of much ongoing research in the field of public and occupational health and safety.

According to Dr. Renaud Vincent, a researcher with Health Canada, even the relatively low levels of diesel exhaust emissions found in urban and occupational settings can cause acute and chronic illnesses of the heart and lungs. The harmful, acute effects of diesel emissions observed by Dr. Vincent included increased blood pressure, increased peptides and elevated heart rates, while long-term effects included brain inflammation and an alzheimers-like pathology to those exposed for longer periods. Reduced neurobehavioral characteristics were also noted in exposed children, as well as lower IQs when compared to their peers who were not exposed to the emissions (Vincent, 2011).

Research in the field is ongoing, and links between diesel engine emissions and a variety of acute and chronic illnesses, (e.g., cardiovascular effects, reduced pulmonary function, allergic response and upper and lower respiratory tract irritation) are continuing to be discovered.

# 2.3 Heat

Diesel engines likely to be encountered in mining equipment can be expected have a thermal efficiency between 30 and 40%.  For every kilowatt (kW) of useful work produced, as much as 3 kW of heat is produced.  Of this heat, approximately one-third is released in the form of exhaust gases, one-third is directly radiated from the machine, and the remainder is produced as frictional heat (the portion of shaft output power less the Work done) (McPherson, 2009).

The heat produced by diesel equipment is also somewhat unique in the mining environment due to the fact that a significant portion of the ehat produced is latent heat.  Latent heat is produced directly in the exhaust gases as water vapor, from emissions controls (e.g., SCRs), and evaporation.  Typically, an operating piece of diesel equipment produces from three to ten liters of water for each liter of fuel used.

Figure 2.2 illustrates the breakdown of motor power being transferred to heat and Work.

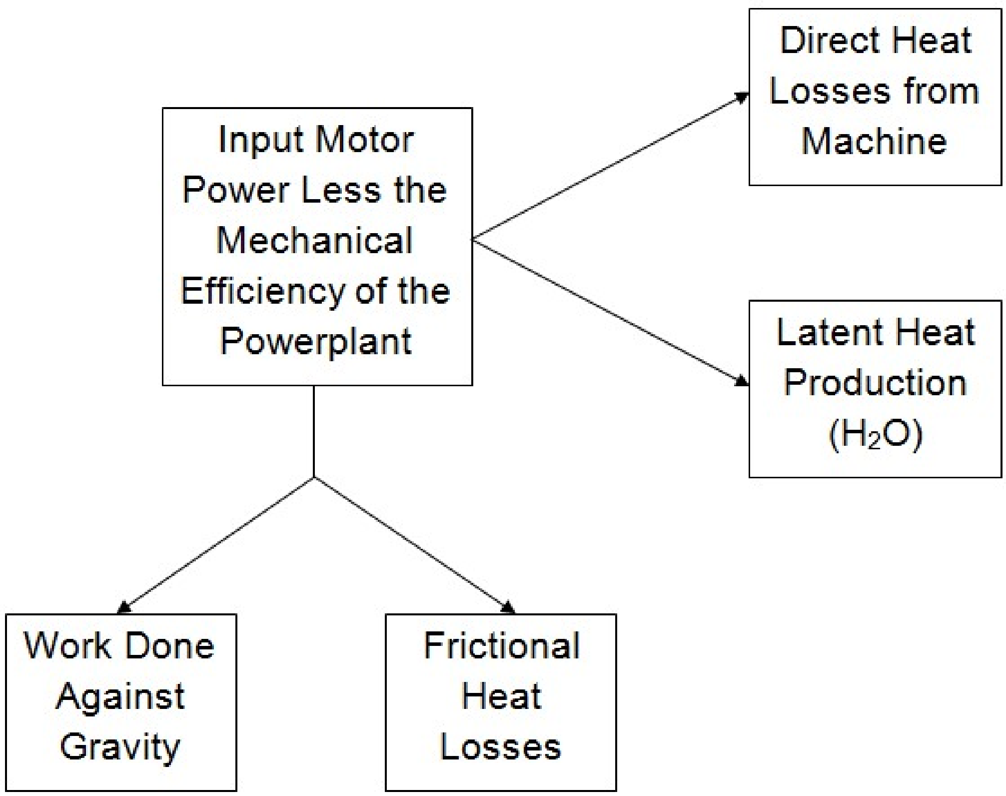


**Figure 2.2: Heat production of diesel engines by type/mode.**

Although the terms are often used interchangeably (incorrectly so), *heat stress* is an environmental condition of heat, while *heat strain* refers to the physiological response to the heat of the human body.

There are many methods for measuring heat loading and heat stress, with some being more applicable for use in underground mines than others.  Some of the most commonly utilized methods will be presented here.

The ACGIH utilizes wet-bulb globe temperature (WBGT) as the metric for establishing TLVs or other action levels based on heat stress.  In cases where a radiant heat source is not visible (i.e., underground mines), the WBGT is governed by the wet-bulb (WB) temperature and he globe temperature of the air surrounding the globe (a non-reflective, hollow black sphere).

The recommended TLVs and action limits from the ACGIH are presented in Table 2.1 based on acclimatized workers (workers who have spent at least five of the past seven, or seven of the last 10 days working the defined conditions adjusted for clothing type and rate of work).

**Table 2.1: TLV and Action Limit for heat stress exposure (ACGIH, 2007).**

| Allocation of Work in a Cycle of Work and Recovery | TLV (WBGT values in °C) | | | | Action Limit (WBGT values in °C) | | | |
|---|---|---|---|---|---|---|---|---|
| | Light | Moderate | Heavy | Very Heavy | Light | Moderate | Heavy | Very Heavy |
| 75% to 100% | 31.0 | 28.0 | N/A | N/A | 28.0 | 25.0 | N/A | N/A |
| 50% to 75% | 31.0 | 29.0 | 27.5 | N/A | 28.5 | 26.0 | 24 | N/A |
| 25% to 50% | 32.0 | 30.0 | 29.0 | 28.0 | 29.5 | 27.0 | 25.5 | 24.5 |
| 0% to 25% | 32.5 | 31.5 | 30.5 | 30 | 30.0 | 29.0 | 28.0 | 27.0 |

The WBGT has also been adopted as the metric for thermal environmental measurements by NIOSH (NIOSH, 1986) and the International Standard (ISO, 1982); however, predicting the WBGT based upon known environmental inputs is difficult and the accuracy of the results is often insufficient for planning purposes.

For ventilation system modeling and design, it is often useful to adopt another climatic parameter for determining the heat load of a given system (e.g., wet-bulb temperature).

The physiological responses and negative health effects of heat stress can vary from mild discomfort to death, with a complete spectrum of potential between the two extremes.

Figure 2.3 illustrates the range of physiological responses to heat stress.

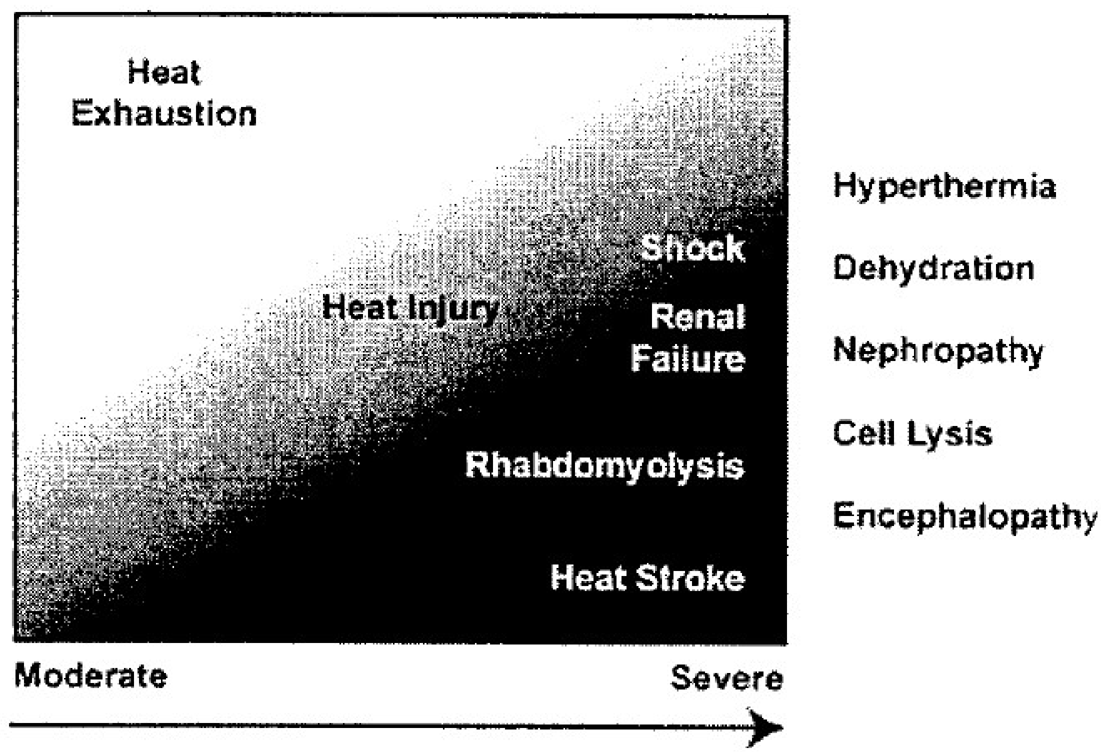


**Figure 2.3: Spectrum of heat strain in humans (US Army RIEM, 2003).**

It is also important to note that individual mine workers will also very in their tolerances to heat stress and their acclimatization levels.  All this adds up to make it extremely difficult to predict just how the climatic conditions of a mine may affect the workforce.

Nonetheless, ventilation system designers often find it is necessary to make predictions of mine climate, and establish criteria which affect the exposure of mine workers to heat stress underground.

The two greatest factors that affect heat strain are the metabolic (work) rate of an individual and the type of clothing (insulation) they are wearing.  Most mine workers are engaged in heavy work while wearing relatively heavy (protective) clothing.  In addition, the body's cooling ability may be affected by a host of other factors, including dehydration, salt depletion, lack of acclimatization, poor physical fitness, excessive body weight, disorders of the skin or endocrine systems, fever, gastro-intestinal diseases, heart disease, diabetes, genetic disorders and drug and alcohol use (US Army REIM, 2003).

Heat stress can also negatively impact cognitive function and the body's ability to perform routine tasks under some conditions (Hardcastle, 2012).  In practice, these mental responses to heat stress are notoriously difficult to quantify, however, this topic is the subject of ongoing studies in the field.  It is currently thought that accident prevention and worker productivity can be improved through the prevention of heat strain in underground workers.

Heat strain prevention underground is accomplished through a dynamic combination of staff training, workforce monitoring and implementing effective controls.  Employees should be trained how to identify heat strain and implement an effective action plan that is immediately implemented whenever heat strain is suspected.

Time should be taken to acclimatize workers to their work environment and activities, and potentially aggravating factors (e.g., fitness, genetic predisposition, etc.) should be identified and corrected as possible.

All workers should be given adequate supplies of water, ice and electrolytes as necessary.  It may also be necessary to restrict work activities to allow for periods of rest and recovery time (e.g., work 45 minutes, rest 15 minutes).

Finally, it may become necessary to dress workers in lighter-weight wicking work clothes, to isolate workers from the environment utilizing air-conditioned vehicle cabs or to consider various bulk air cooling methods such as spot chillers or surface refrigeration.

# 2.4 Dusts

Even though it is not directly emitted from diesel engines in the way that gases, particulates and heat are, almost all diesel equipment operating in mines will produce mineral dust when they are operated in underground environments. This dust generated by operating diesel equipment is therefore included in the list of contaminants that must be considered by ventilation engineers and others involved in the design, measurement and maintenance of underground ventilation systems.

The dust(s) commonly encountered in mines are typically classified by their particle size (i.e., respirable, non-respirable) or by their composition (e.g., silica, coal, asbestos, etc.).

Particle size classifications are often made equating the dust particle to the volume of a sphere with an *equivalent geometric diameter*.

The respirable fraction of the dust refers to those particles which are small enough to be able circumvent the body's natural defenses reach the deep lung tissue and be quickly adsorbed by the pulmonary system. Respirable dust ranges in size from sub-micron particles up to seven microns.

Respirable dust represents as significant health hazard to mine workers owing to their ability to stay in suspension almost indefinitely and to be inhaled and deposited into the lungs where their tiny scale allows them to more easily react with the body's tissues and chemistry.

Although their small size renders respirable dust particles invisible to the naked eye, it may be assumed that they are present in any area where visible dust is observed (McPherson, 2009).

There are many acute and chronic illnesses than may result from exposure to dust commonly encountered in underground mines. They range in seriousness from mild annoyance to death.

McPherson separates the types of dust into five categories, based on the particular health effects produced by each group. They are:

1. Toxic Dust, which comprises dust that cause detrimental chemical reactions in the body once inhaled (e.g., arsenic, cadmium, lead, nickel, selenium, etc.).
2. Carcinogenic Dust, which consists of dust that stimulates the growth of tumors in the lung tissue (e.g., asbestos, DPM, quartz, radon daughters, etc.).
3. Fibrogenic Dust, consisting of dust that causes the formation of scar tissue in the lungs (e.g., asbestos, quartz, mica, talc).
4. Explosive Dust, comprising dust that it is possible to detonate (e.g., coal, sulphides and other metallic dust).
5. Nuisance Dust, which comprises those dust types that result in minor, acute health effects such as irritation of the eyes and upper respiratory tract, but do not lead to negative chronic conditions (e.g., gypsum, halites, potash, etc.).
6. Although their small size renders respirable dust particles invisible to the naked eye, it may be assumed that they are present in any area where visible dust is observed (McPherson, 2009).

The individual geology of a particular mining environment is the most influential factor in determining what type(s) of dust will be encountered in any given mine. All though it is possible to find all five dust types in underground mines (some mines can have all five present at the same time), the most commonly encountered dust types fall into the fibrogenic and nuisance categories.

When inhaled and deposited into the lungs, all types of dust with the exception of those that fall in to the category of nuisance dust can lead to serious chronic health problems. In some cases, they may act in combination and exacerbate the relative harm that each would do individually. Scars are cut into the lung tissue by fibrogenic dusts' sharp and hard particles, reducing lung capacity over time. Coal dust clots together in the lungs to form barriers to oxygen transmission and reduce lung function. Tumors formed in the lungs by carcinogenic dust may be metastasized and spread throughout the body.

Chronic health illnesses resulting from dust exposure have long been recognized within the mining industry, with the first legislation appeared to govern the gold mining that was occurring in the Witwatersrand of in South Africa in 1912 (McPherson, 2009).

Current TLVs for dusts commonly encountered in suspension as recommended by the ACGIH are given in Table 2.2.

**Table 2.2: TLVs for Selected Aerosol Compounds (ACGIH, 2007).**

| Substance (Documentation Date) | Adopted Values TWA | STEL | Notations | MW | TLV Basis |
|---|---|---|---|---|---|
| Aluminum oxide | 10 mg/m$^3$ | - | A4 | 102.0 | LRT irr; pneumoconiosis |
| Arsenic (1990) | 0.01 mg/m$^3$ | - | A1; BEI | 74.9 | Lung cancer |
| Asbestos (1994) | 0.1 f/cc | - | A1 | N/A | Pneumoconiosis, mesothelioma |
| Calcium silicate (1988) | 10 mg/m$^3$ | - | A4 | - | URT irr |
| Calcium sulphate (2005) | 10 mg/m$^3$ | - | - | 136.1 | Nasal symptoms |
| Carbon black (1985) | 3.5 mg/m$^3$ | - | A4 | - | |
| Coal anthracite (1995) | 0.4 mg/m$^3$ | - | A4 | - | Lung dam; pulm fibrosis |
| Coal bituminous (1995) | 0.9 mg/m$^3$ | - | A4 | - | Lung dam; pulm fibrosis |
| Flourides (1979) | 2.5 mg/m$^3$ | - | A4; BEI | Varies | Bone dam; flourosis |
| Graphite (1988) | 2 mg/m$^3$ | - | - | - | Pneumoconiosis |
| Kaolin (1990) | 2 mg/m$^3$ | - | A4 | - | Pneumoconiosis |
| Magnesium Oxide (200) | 10 mg/m$^3$ | - | A4 | 40.3 | |
| Mica (1962) | 3 mg/m$^3$ | - | - | - | Pneumoconiosis |
| **Nuisance dusts** | | | | | |
| Oil Mist - mineral (1992) | 5 mg/m$^3$ | 10 mg/m$^3$ | (-) | - | (Lung) |
| Portland Cement (1992) | 10 mg/m$^3$ | (-) | (-) | - | (Irr; dermatitis) |
| Radon Daughters | 4 WLM/year | - | - | - | |
| Silica crystalline $\alpha$ Quartz (2205) | 0.025 mg/m$^3$ | - | A2 | 60.1 | Pulmonary fibrosis; lung cancer |
| Silicon Carbide (1999) | 10 mg/m$^3$ | - | - | | URT irr |
| Talc (1980) | 2 mg/m$^3$ | - | A4 | - | LRT irr |
| Zinc Oxide (2001) | 2 mg/m$^3$ | 10 mg/m$^3$ | - | 81.4 | Metal fume fever |

Although great progress in identifying and combating dust-related illnesses has been made, diseases such as coal-workers' pneumoconiosis or "black lung", silicosis, mesothelioma and other bronchial cancers are continuing to debilitate and kill workers exposed to various dusts in mines around the world.

However, the prevention of these diseases can be accomplished by the mining industry provided that adequate knowledge and resources are invested in implementing a variety of engineering and administrative controls. These dust prevention and reduction measures, when paired with an effective education and monitoring program for potentially exposed workers can be successful in completely eliminating dust-related illnesses from the mining industry.

# 3.0 Measurement of Diesel Emissions

Sampling forms the foundation for a comprehensive understanding of the level and type of diesel emissions exposure experienced underground.  Ambient air sampling allows mine operators to assess the effectiveness of their emissions control and reduction program and identifies whether or not they are in compliance with applicable regulations and standards.

Contaminant levels vary from mine to mine and among different locations within the same mine, meaning that even educated assumptions with regard to contaminant levels are insufficient once a mine has begun operation.  A comprehensive sampling program should cover all tasks and areas of the mine in order to ensure that no potential hazards or hazardous conditions are missed.

# 3.1 Gaseous Emissions

Ambient measurements of gaseous tailpipe emissions from diesel equipment are typically measured with chemically-reactive “stain tubes” or with a digital gas sensor that may be handheld or fixed in place and which may or may not be connected to a central monitoring network.

Gas detection tubes that are manually aspirated with a small rubber bulb represent the cheapest, albeit least accurate option for measuring ambient gas concentrations. They are specific to a specific gas, and will produce only a single reading per tube. Air is sucked into the tube via a small hand pump or bulb, and the gas concentration is displayed via a scale on the side of a tube in ppm. This process provides a direct reading of gas concentration in real-time, at a relatively low cost.

Figure 3.1 gives an example of a gas-detecting “stain tube” with some explanation.

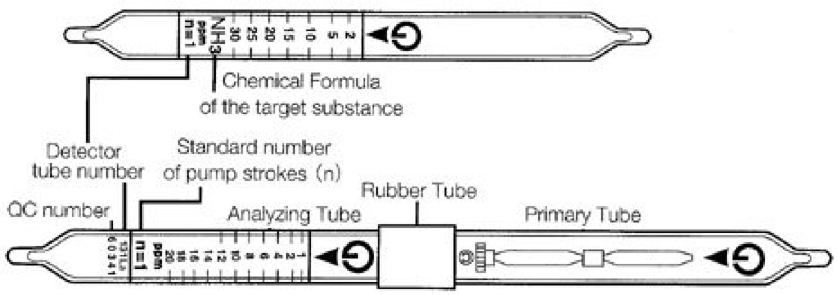


**Figure 3.1: Gas detection tube (Environmental Equipment and Supply).**

Hand-held digital gas detectors are probably the most popular means for conducting measurements of $NO_x$, CO and $CO_2$ (as well as other gases) in mines. These electro-chemical devices can determine the concentrations of various gases by generating an electrical charge in the presence of that gas. The intensity of the charge is then converted to a digital signal that is displayed on the screen of the device.

Similar devices have been designed to be fixed in place and provide continuous gas concentrations (or at regular intervals) that can be displayed on a digital readout or sent to a central monitoring location and recorded.

Like any digital sensor, these gas detectors require periodic maintenance and calibration in order to function properly. Despite their high cost (relative to the stain tubes), their accuracy, precision and durability have made them the standard for measuring underground gas concentrations.

# 3.2 Particulate Emissions

The most accurate method for measuring ambient levels of DPM is the NIOSH 5040 method.  This method specifies the use of a small, constant-volume pump that pulls air through a sampling train (including a cyclone and/or a jeweled-impactor cassette) designed to remove all particles greater than one micron prior to the deposition of DPM on a quartz-fiber filter.

When collecting ambient DPM sampling, detailed and accurate records and observations are critical to understanding the results. The factors affecting DPM exposure in an underground environment are many and varied, and often act in concert to create high exposure levels.

In addition to notes recording the pump information (pump no., type, flow rate, etc.), sample no., and personnel or area information where the sampling in conducted it is necessary to observe conditions throughout the sampling period and record information regarding the operating equipment, visibility, airflow quantity and velocity and other observations.  This information will be critical in understanding how the DPM value calculated in the laboratory relates to the specific conditions in the mine that contributed to that level of exposure.

As the calculations required for calculating the final DPM exposure are heavily dependent on the total volume of flow across the filter, it is necessary to calibrate the pump flow-rate before and after the measurement is taken.

Once the sample is taken, a portion of the quartz-fiber filter is taken with a filter punch that carefully removes an exact area of the filter.  The filter punch is placed into the thermal optical analyzer and run through a computer-controlled sample analysis in two stages.

Stage I involves increasing the temperature of the oven gradually up to 850 °C in a helium atmosphere.  The carbon present on the filter is catalytically oxidized to $CO_2$ and eventually reduced to methane in a methanator made of nickel-firebrick and measured by a flame ionization detector.

Stage II of the process flushes the oven with a mixture of oxygen and helium while increasing the temperature up to 900 °C.  During this stage the pyrolytic carbon is oxidized and the laser transmittance through the filter is increased.  Any increase in the laser transmittance once it reaches its initial value is considered to be Elemental Carbon.

From the time the filter is placed into the machine to the time it is removed, the entire process is automated and controlled by the computer, including the determination of Organic and Elemental Carbon and a record of the analysis being saved to a data file.

Before any samples are analyzed, the analyzer is calibrated by running a series of tests to determine the machine’s accuracy and precision.

Calibration standards are filters that have a known amount of Carbon deposited on them.  The measured amount is compared with the known amount to verify that the machine is displaying and recording the correct values.

The zero-point of the machine is tested by running a blank filter punch through an oven-cleaning process designed to burn off any Carbon present.  This “oven-cleaned” sample is then fully analyzed to ensure that the machine returns a zero-value for Carbon.

Duplicate samples are also run periodically, on a minimum of ten percent of samples analyzed to determine sample precision.

The thermal optical records the fractions of Organic and Elemental Carbon present on the sample as well as the ratio between the two values and the Total Carbon present (the sum of the Organic and Elemental fractions).  The values are reported in units of micrograms of Carbon present per square centimeter of filter.  The relevant value of Carbon content (usually EC or TC) is multiplied by the total filter deposition area in order to determine the total amount of carbon present on the filter.

Once the Carbon present on the filter is known, the determination of DPM exposure may be calculated by dividing the amount of Carbon by the total volume of air pulled across the filter.

The DPM exposure is calculated according to the following equation:

Equation: $$E = \frac{TC(Qp)t}{1000}$$

here: $E$ = DPM exposure ($\mu g/m^3$)

$TC$ = total carbon ($\mu g$)

$Q_p$ = pump flow-rate (l/min)

$t$ = pump run-time (min)

The NIOSH 5040 method represents the most accurate determination of DPM possible, and is specified by the MSHA regulations governing compliance testing.

Similarly, to any laboratory process that is evaluated for causative effects using statistical methods, it is important to follow a rigorous quality control program for the entire sampling process, including during sample collection, storage, laboratory analysis and during any statistical analyses that follow.

The first real-time units to offer the measurement of DPM exposure in real-time, that is, an extrapolation of instantaneous readings into a reading of DPM exposure in micrograms per cubic meter were belt-wearable devices capable of determining the amount of respirable combustible dust that passed through the unit utilizing a photo-accoustic or condensation counter. However, these early real-time DPM monitors suffered from an inability to accurately differentiate between DPM and other common aerosols in the mine environment, like mineral dust and oil mist.

In response to this deficiency, a new real-time DPM monitor was developed by the Respiratory Hazards and Control Branch at NIOSH. The device utilizes measurements of laser transmittance (similarly to process used by the NIOSH 5040 method but without the need to heat the samples). Air is sucked into the unit via a particle separator where anything larger than one micron is discarded. The submicron particles are collected on a replaceable filter, causing changes to the transmittance of an internal laser. The changes in laser transmittance are recorded and the values for the equivalent DPM exposure are displayed. This determination is accomplished through the use of an internal algorithm that was developed directly from data obtained by analyzing samples via the NIOSH 5040 method.

One great advantage of real-time monitors is that they allow an analyses of when the DPM exposure is actually greatest during the sample period (work shift), leading to the rapid correlation between exposure levels and their contributing conditions (e.g., a specific piece of equipment or a deficient auxiliary ventilation system.

An example of a real-time DPM monitor showing a sample readout is shown on Figure 3.2.

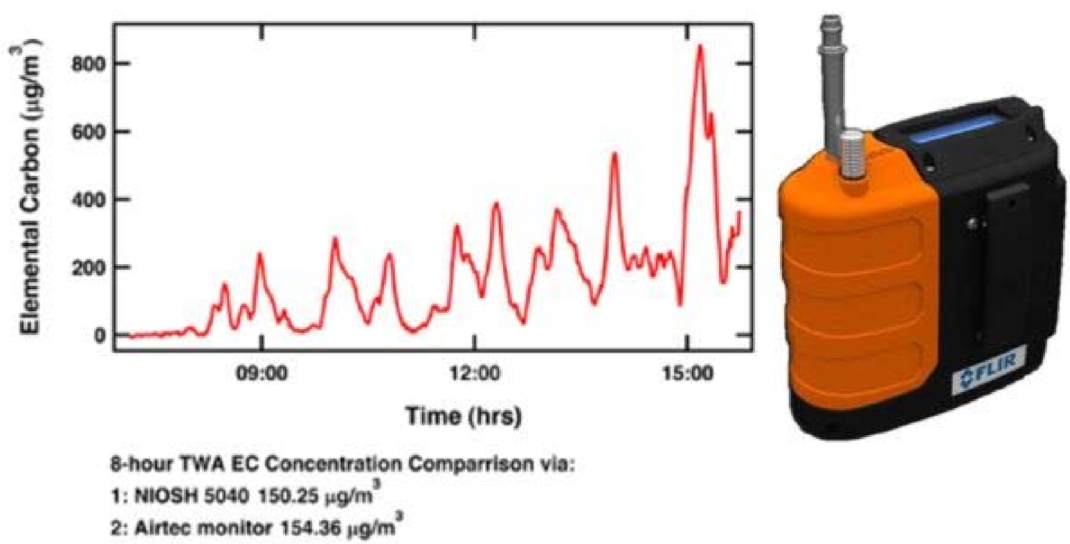


**Figure 3.2: Real-time personal DPM monitor manufactured by FLIR.**

# 3.3 Heat

Sampling thermal parameters to identify heat stress in the mine should be conducted as often as necessary as part of a regular program for environmental health. As the criteria for heat stress and heat management will vary between companies, the sampling protocol will likewise change.

Thermal sampling (a temperature survey) should include all areas of the mine where personnel work or travel; however, measurements should be concentrated in areas where the risk of heat strain in personnel is most acute.

Measurements may be taken with portable devices that are carried to different areas of the mine, via sensors that are fixed in place and log values in a central location or both. Most mines will utilize some combination of sensors and measurements in order to gain the best possible results.

Mine weather stations installed at key locations underground can provide real-time data to a central monitoring location in addition to logging this data for future analyses, while portable units can be mobilized to remote and varied areas of the mine as needed.

One key decision that must be made when designing a system for the measurement of climatic parameters is what values to measure (e.g., WBGT, $T_{WB}$, $T_{DB}$ and Relative Humidity (RH), etc.). This decision should be based on the application of said data.

If the purpose of the measurements is to identify compliance with a company or regulatory standard for heat stress, then WBGT should be measured (assuming that this is the value specified in the standard). However, if the measurements are being conducted for the purpose of validating a climatic model then it may be more appropriate to measure $T_{WB}$ or $T_{DB}$ and RH.

A diverse selection of permanent and hand-held sensors and devices are available for mining and underground tunneling applications. Their performance, as well as their price varies greatly, so it is advisable to do some research and comparison prior to making a choice of which unit(s) to use.

Figure 3.3 shows some examples of instruments for measuring climatic parameters.

**Figure 3.3: Portable WBGT-measurement tool (General, 2011).**

# 3.4 Dust(s)

Dust sampling in underground mines is both difficult and necessary. The sampling methods and protocols for measuring the dust generated by diesel equipment underground is fundamentally no different than measuring for dust generated by any other source.

In some cases, the type of sampling equipment or the locations to the sampled may be governed by regulatory bodies, whereas personal samples are simply attached to the selected worker.

Typically, a Dorr-Oliver type cyclone is used in conjunction with a constant volume pump with a flow rate set to the desired cut rate in order to select the appropriate particle size for respirable dust sampling (e.g., 4 micron, 5 micron, etc.).

A cross-section of a typical dust classification cyclone is shown on Figure 3.4.

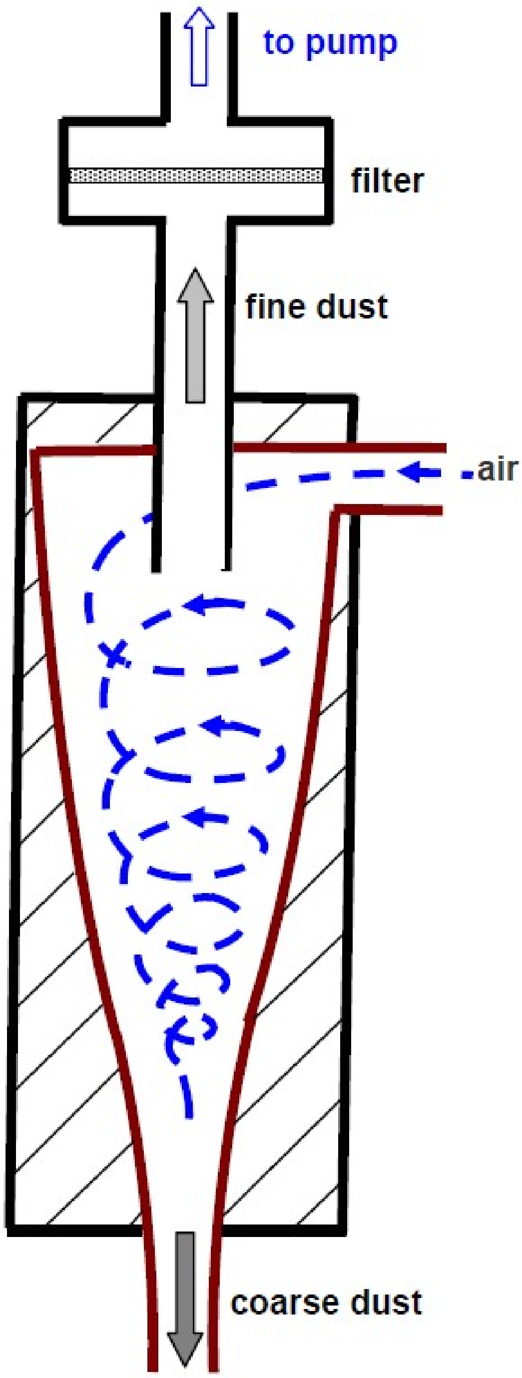

**Figure 3.4: Cross-section of a dust classification cyclone (McPherson, 2009).**

According to 2010 NIOSH report, gravimetric sampling represents the most common method for measuring ambient levels of dust in underground mines. Both the sampling method and apparatus are similar to that utilized in for the collection of DPM samples in accordance with the NIOSH 5040 method.

Figure 3.5 depicts a gravimetric dust sampling pump and sampling train (filter and cyclone).

**Figure 3.5: Typical Gravimetric Dust Sampling Train (NIOSH, 2010).**

Air is pulled through a cyclone or other dust classification device by an adjustable-rate, constant volume pump with the flow-rate set to achieve the desired cut rate of particles. The respirable dust fraction is collected on a filter of known weight, which is then re-weighed on a precise scale to obtain the total mass of dust.

The dust concentration is then calculated by dividing the total sample weight in micrograms by the volume of air passed through the filter in cubic meters.

The silica content of a particular sample may be determined through x-ray diffraction (XRD), requiring a specialized laboratory/equipment.

If real-time measurement of the dust concentration is required, these are also possible through a series of devices that utilize various methods to determine the dust content present in the ambient environment.

Photometric sampling devices determine the dust concentration by passing a beam of light through an airstream and measuring the deviation (scattering) of the light which is then correlated to a specific concentration of dust present. These devices are susceptible to interference from aerosol water and oil droplets, however.

Figure 3.6. provides a schematic of a real-time photometric dust sampler.

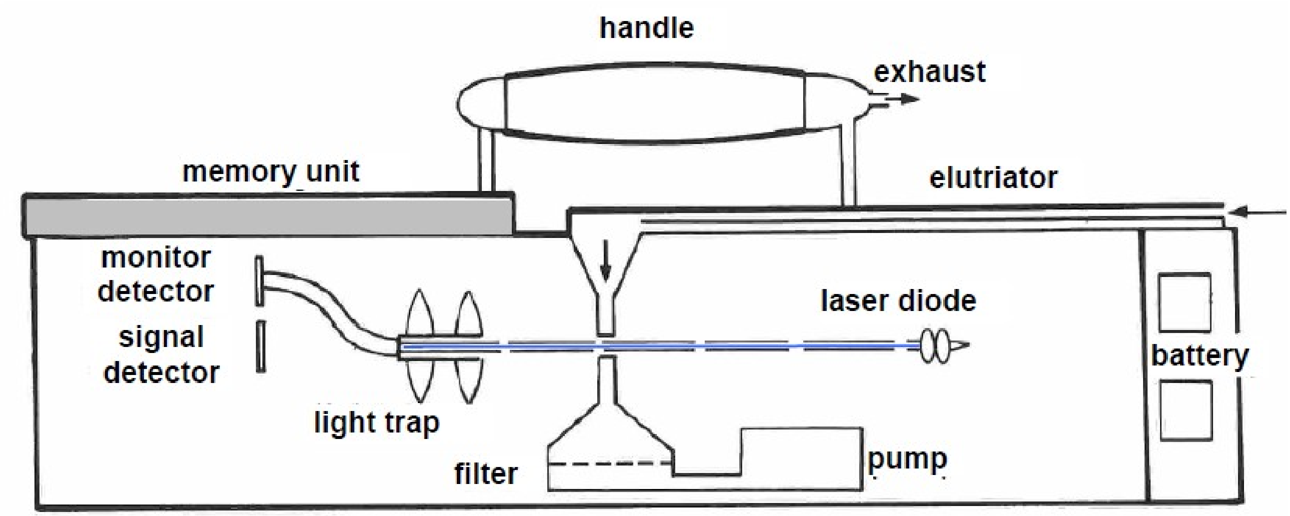


**Figure 3.6: Schematic of a real-time photometric dust sampling device (McPherson, 2009).**

A more common personal dust monitor (PDM) was developed by NIOSH and has been in use in underground mines since 2006. This PDM utilizes a tapered-element oscillating microbalance (TEOM) to give gravimetric measurements of dust in real time. The TEOM PDM includes a dust filter on the end of a tube that oscillates at a known frequency. The buildup of dust on the end of the pendulum changes the frequency of oscillation, which can then be correlated to the ambient concentration of dust. The dust exposure measured by the unit is displayed on the screen readout in real-time and is also logged in to the device's memory for later recall. The ability to download a digital record of the exposure to a central computer or database for analysis and recordkeeping is a distinct advantage to this device.

Figure 3.7 shows the fundamental components of the NIOSH PDM including the TEOM module.

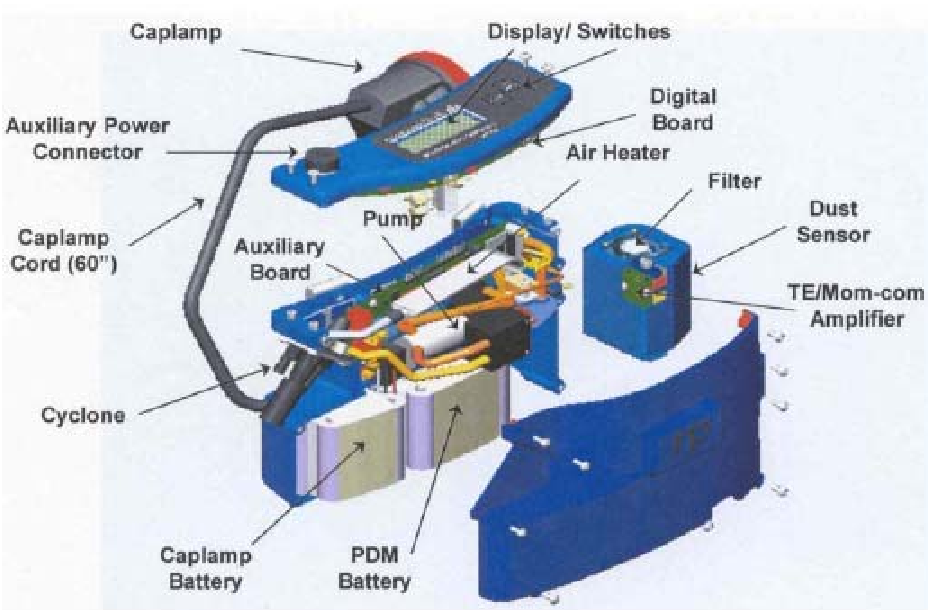


**Figure 3.7: Exploded view of the NIOSH PDM (McPherson, 2009).**

# 4.0 Diesel Emissions Control (Reduction)

A wide variety of control options for diesel emissions and contaminant products exist.  All have slightly different effects and interactions.  When developing a plan for diesel emissions control, it is important to consider the whole system, and utilize a combination of control methods and technologies that work well together.

# 4.1 Ventilation

The most basic method for controlling diesel emissions in underground mines and facilities is by diluting the gases and particulates with fresh air that is circulated over the equipment by the primary or auxiliary ventilation system(s).

The primary ventilation system includes all areas of the mine that receive “flow-through” ventilation, or have communication with more than one other branch of flow.  Airflow through this network is induced via in-line fans (primaries and/or boosters) and distributed via passive ventilation controls such as walls, doors and regulators.

The total flow quantity in the primary ventilation circuit must be sufficient to mitigate all contaminants present in the underground (e.g., dust, heat, gases, DPM, etc.) from all sources, and may be limited by the physical dimensions of the mine infrastructure or other parameters that form the *basis of design* (BOD) for the project.

Auxiliary ventilation systems are used to ventilate areas that fall outside of the primary ventilation circuit (i.e., dead-end tunnels and excavations) and likewise are limited by physical mine parameters, since sufficient clearance must be maintained between ducts and any other mine equipment or services that are present.

Auxiliary ventilation systems are further constrained by the amount of air present in the primary ventilation circuit, as if the quantity circulated exceeds that of the airflow provided by the primary circuit, then they may actually contribute to the build-up of contaminants, rather than their reduction.

Regardless of whether the fresh air required for contaminant dilution is supplied via the primary or an auxiliary ventilation system, the relationship governing the reduction in contaminants is as follows:

**Equation:** $$C = \frac{Gf}{Qf}$$

where: $C$ = contaminant concentration (%)

$G_f$ = contaminant flow-rate

$Q_f$ = ventilation flow-rate

This simple equation allows the rapid determination of any airflow (or increase in airflow) quantity that is required to achieve an acceptable level of the contaminant in question.  The simplicity and reliability of this method for reducing airborne contaminants is the reason for its popularity, but it does have very specific limitations with regard to the upper limits of airflow quantity required, and increases to the maximum capacity of the primary ventilation system are often expensive and time consuming (e.g., new ventilation shafts, slashing existing drifts/raises, or parallel entries, etc.).

# 4.2 Fuels

The type of diesel fuel consumed by the equipment will have an effect on both the quantity and quality of the emissions produced at the tailpipe (gases and particulates). The type and quality of the fuel may also affect the emissions control strategy, as not all reduction technologies work with all fuel types and qualities. In general, the more advanced the engine technology (i.e., Tier IV), the more refined and consistent fuel is required.

Since 2014, all diesel fuel sold in the US must meet the standard for Ultra-Low Sulfur Diesel (ULSD) which has a fuel sulfur content less than 15 ppm. Although this is now the standard for most diesel fuel, users in the developing world should note that ULSD is not always available, depending on geographical and economic factors unique to individual sites.

Figure 4.1 shows worldwide limits for fuel sulfur content where such limits are in place (or enforced).

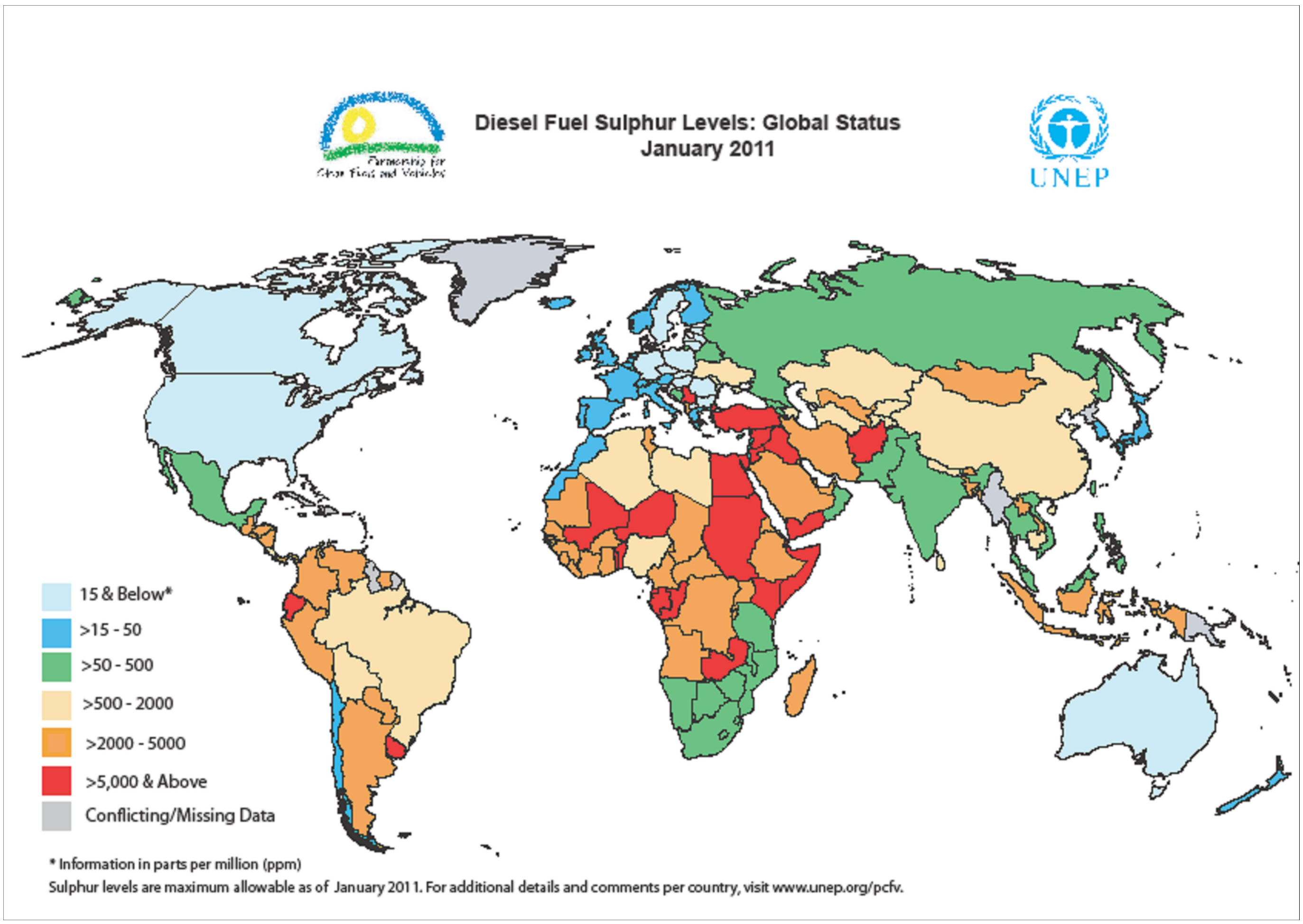


**Figure 4.1: Diesel sulfur standards around the world (UNEP, 2012).**

Generally, the organic fraction of diesel particulate emissions is reduced in proportion to the reduction in fuel sulfur content (Bugarski, 2011). Perhaps even more critical however, is the fact that many modern diesel engines and emissions-reduction technologies will simply not function with high-sulfur fuels. For example, high-efficiency diesel particulate filters will quickly become plugged when used with high-sulfur fuels, which can damage the engine or cause a fire owing to the increased backpressure in the equipment's exhaust system.

Biodiesel fuels have also been used to reduce tailpipe emissions from diesel equipment operated in underground mines and tunnels. Biodiesel may refer to a wide variety of natural and synthetic substitutes for diesel fuel, including frying oil, and soy-based fuels. The reduction in DPM when using biodiesel fuel is roughly equivalent to the amount of diesel that is replaced (i.e., a mixture of 20% biodiesel and 80% petro-diesel will have a 20% reduction in engine-out particulate emissions.)

Although it may be possible to utilize a fuel composed of 100% biodiesel, these fuels are typically mixed with petro-diesel in order to minimize any potential negative impacts to the reliability and life of the engine and fuel system components. Often, the use of biodiesel blends greater than B-20 (20% biodiesel) will void any existing manufacturers' warranty on the equipment in question.

# 4.3 Emissions-Based Maintenance Programs

The implementation of a maintenance program specifically designed to reduce emissions from diesel equipment has been shown to be one of the most effective methods of controlling tailpipe emissions (gases and particulates) from operating equipment.

It is important to note that programs of this type are much more involved than typical “preventative maintenance” programs that are implemented solely for the purpose of reducing down-time and prolonging equipment life.

Emissions-based maintenance programs incorporate emissions-specific monitoring and control protocols that monitor and record the emissions of specific equipment over time, and also include components of the equipment’s cooling, intake, exhaust and fuel-delivery systems that all can impact the emissions produced.

These specific, emissions reduction programs have been shown to reduce up to 50% of DPM and 60% of the CO produced by operating diesel equipment (Forbush, 2001).  In addition, many mines have achieved improved fuel economy, engine life and reduced equipment down-time (McGinn, 2000).

# 4.4 Engine Type/Specification

The selection of the engine/powerplant for a particular piece of equipment is the most significant decision with regard to the emissions contribution that can be expected.

Although Tier IV rated diesel engines have significantly reduced gaseous and particulate emission rates, the dust and heat they produce is similar to their precursors.

In existing mines, older equipment that pre-dates Tier IV technology (and in some cases, may even be pre-Tier) must be properly identified and ventilated.

In the US, MSHA publishes detailed emissions data including the particulate index and the grams per horsepower-hour for each diesel emission approved for use underground.

In Canada, approved diesel engines are assigned an Air Quality Index (AQI) that is based on the amount of air required to dilute the exhaust emissions.

Although the diesel engine is still the undisputed “king” of the mining equipment powerplants, advances in technology coupled with the rising cost of power and infrastructure required for ventilation have made the case for several alternatives, including liquefied natural gas, electric, diesel-electric (hybrid) and fuel-cell powered vehicles.

Diesel-electric hybrid powerplants can realize 40-60% reductions in exhaust emissions, while zero exhaust emissions are produced by vehicles powered by fuel cells (Matikainen, Hodgins, 2010).

Although electric-powered mining equipment do not produce exhaust gases or particulate emissions, they still generate heat and dust when they are used underground, and still require ventilation to operate safely. The determination of how much airflow is required is performed in a similar manner as when calculating the total quantity required to mitigate the heat and dust created by diesel equipment, as it is not possible to determine the required airflow from a single numeric factor (Willick, 2010)

As technology advances, and other parameters change the “Basis of Design” for underground mining projects, alternative power sources for mining equipment may become a significant consideration in mine ventilation system(s) design.

### Case Study: Onaping Deep Mine

Onaping Mine, located in the province of Ontario, Canada lies in the famed nickel rim surrounding the city of Sudbury. Although the climate surrounding the mine is quite temperate (often requiring mine air heating during the winter months, heat will become the driver for ventilation system demand as the mine progresses deeper in the near future. Recognizing this, the mine has investigated alternatives to its current diesel equipment fleet, which represents the single greatest heat source within the mine, and the only significant source that can be easily reduced without greatly impacting mine operations.

Owing to the significant gains in efficiency that can be realized through the use of electric equipment (battery) over their diesel equivalents, the mine performed extensive testing with a battery-powered loader with the goal of establishing the feasibility of implementing a battery-powered fleet in the planned Onaping Deep mine expansion.

Based on the study performed by the mine engineering staff, significant cost savings were shown to be achievable based upon; lower ventilation rates, smaller/fewer raises and drifts, less heat from auto-compression, reduced heat from the mine equipment fleet, and reduced air cooling/refrigeration requirements.

The study shows that for the mine expansion project, a potential capital cost savings of $15M could be realized, with a further $8M per year in realized operating cost savings could be achieved by replacing the diesel equipment fleet with battery-powered electric equivalents.

While changes to battery and equipment technology are advancing rapidly, there are still many challenges that exist for mines that wish to convert their equipment fleets. Challenges with battery-powered mine equipment include; up-ramp hauling, battery charging/change-out stations, cycle times, flexibility and infrastructure requirements and availability.

# 4.5 Exhaust Aftertreatment

A wide variety of passive and active exhaust aftertreatment devices may be fitted to a diesel engine in order to reduce the emissions (gases, particulates, or both). These aftertreatment technologies may be installed singularly or in series, with varying effects.

Diesel Oxidation Catalysts (DOCs) refer to a range of products that are installed in-line to the exhaust system and convert CO and hydrocarbons into $CO_2$ and water vapor. The use of a catalyst allows this conversion to occur at a lower than usual temperature. DPM is generally not affected by these devices, which consist of a metallic or cordierite filter installed in the vehicle exhaust pipe.

The chemical reactions that occur in a typical DOC are shown on Figure 4.2.

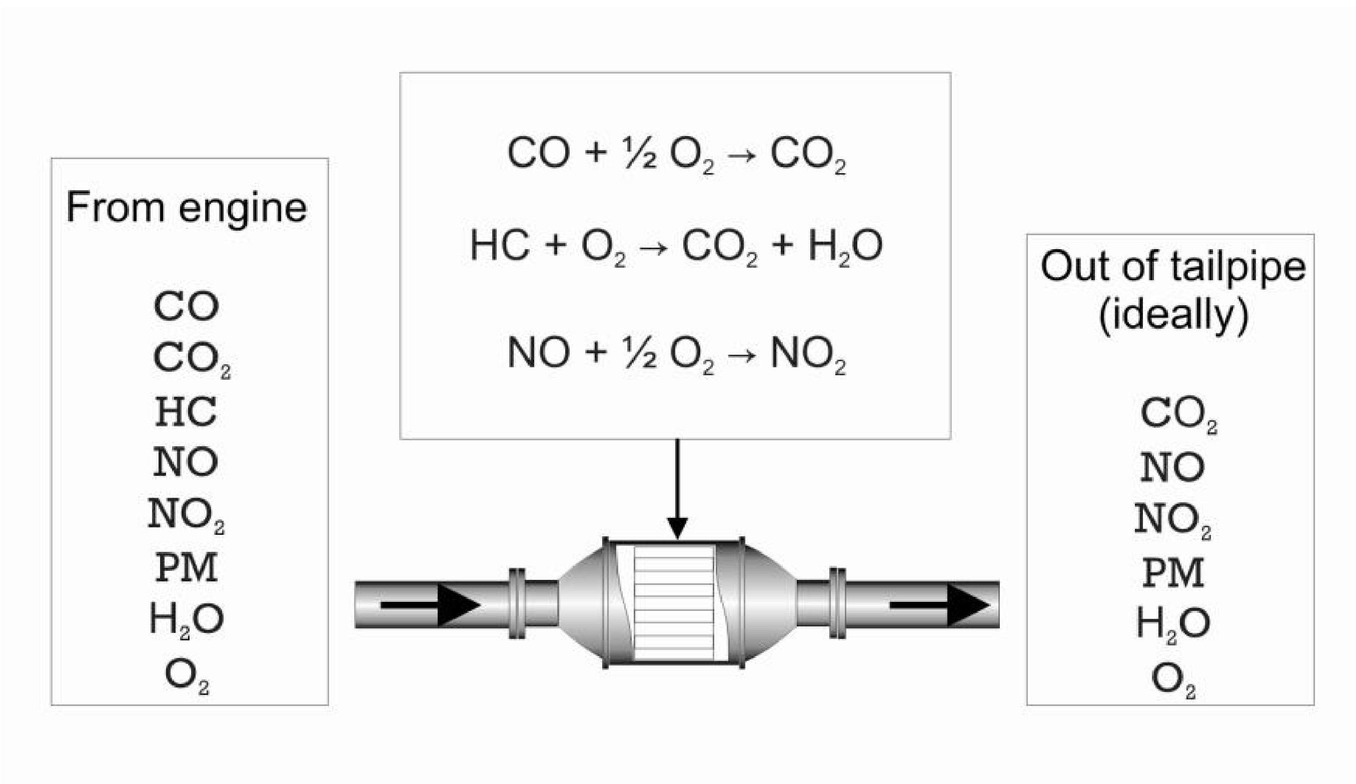


**Figure 4.2: Operational of a Diesel Oxidation Catalyst. (Bugarski, 2011)**

The efficiency of the DOC is largely dependent upon the catalyst that is chosen, generally a base-metal or precious-metal wash that is bonded to the filter substrate. The effectiveness of the various catalysts is the single most important factor in performance, and the difference in effectiveness is wide.

As a result of this variation, and in light of the additional $NO_2$ that can be produced, care should be taken when retrofitting a DOC to existing equipment, but these simple, and relatively inexpensive devices can be utilized to reduce tailpipe emissions, particularly when coupled with other exhaust aftertreatment devices, (e.g., diesel particulate filters) in series.

Diesel particulate filters or DPFs refer to a range of high-efficiency filters that remove particulates from the engine-out exhaust of diesel engines. The term DPF is used to describe a range of products that include filter substrates made from Cordierite, Silicon Carbide and sintered metal.

The filtering process of a wall-flow, ceramic-type DPF is shown on Figure 4.3.

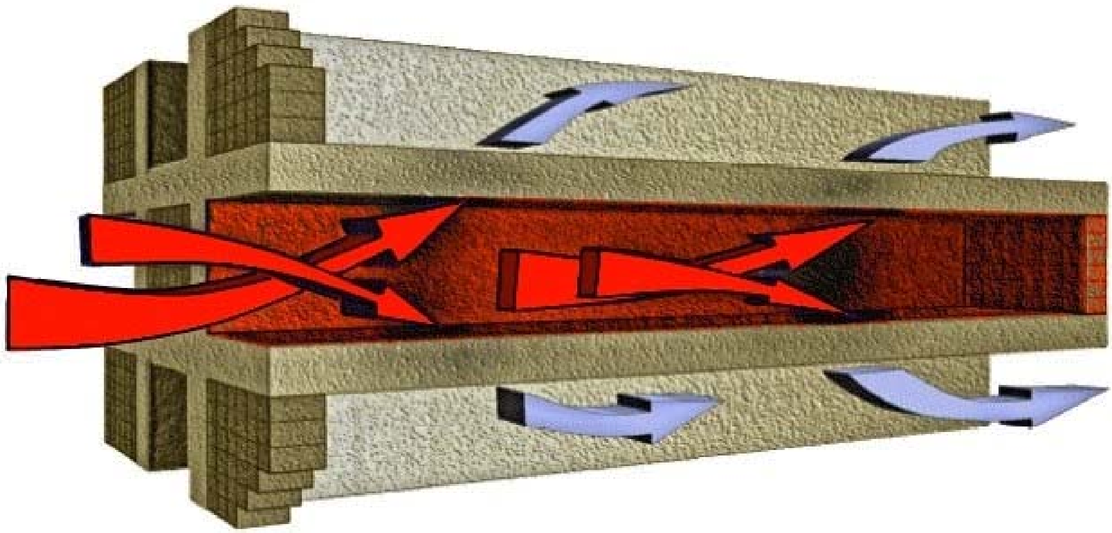

**Figure 4.3: Operation of a standard wall-flow DPF (Diesel Technician Society, 2007).**

The MSHA rates the efficiency of these filters at between 80 and 99% efficient at removing DPM from the exhaust stream, and is helped by the fact that filter efficiency increases proportionally with filter loading.

Actual filter efficiency is dependent upon the condition of the filter substrate as well as a host of environmental factors, such as exhaust constituency, exhaust temperature, presence of any catalysts (base metal, precious metal or other) and filter condition.

Filter substrates may or may not be treated with a catalyst that affects how much, and what is filtered at what exhaust temperatures and what byproducts (e.g., $NO_x$) are created.

As with most devices installed in the exhaust circuit, it is extremely important to size and maintain DPFs correctly, as they can cause severe engine damage and vehicle fires if they become overloaded with filtered material owing to the increase in engine backpressure.

In order to increase filter life and efficacy, most DPFs utilize some method for regeneration (removing the built-up carbon deposited on the filter substrate). Passively regenerating filters are capable of burning off the built up carbon at normal exhaust temperatures, or by periodically injecting fuel directly into the exhaust pipe upstream of the DPF. Actively-regenerated filters require that the filter be removed and "baked" in a specially designed oven. Active regeneration may take a period of 2 – 4 hours to complete, and generally requires the purchase of a second DPF for equipment with high usage so that there is no downtime associated with the regeneration.

For retrofit applications, the selection process for DPFs should begin with a comprehensive monitoring program that logs and analyzes the exhaust temperature and operating conditions of the vehicle(s) in question.

Selective Catalytic Reduction or (SCR) is a process by which urea is injected directly into the engine exhaust stream from an on-board reservoir at a volume of less than 5% of the total fuel consumption. The urea mixture is hydrolyzed into ammonia in an intermediary step and then converted to N2 in the exhaust, reducing NOx during the process.

The chemical processes that constitute selective catalytic reduction are shown on Figure 4.4.

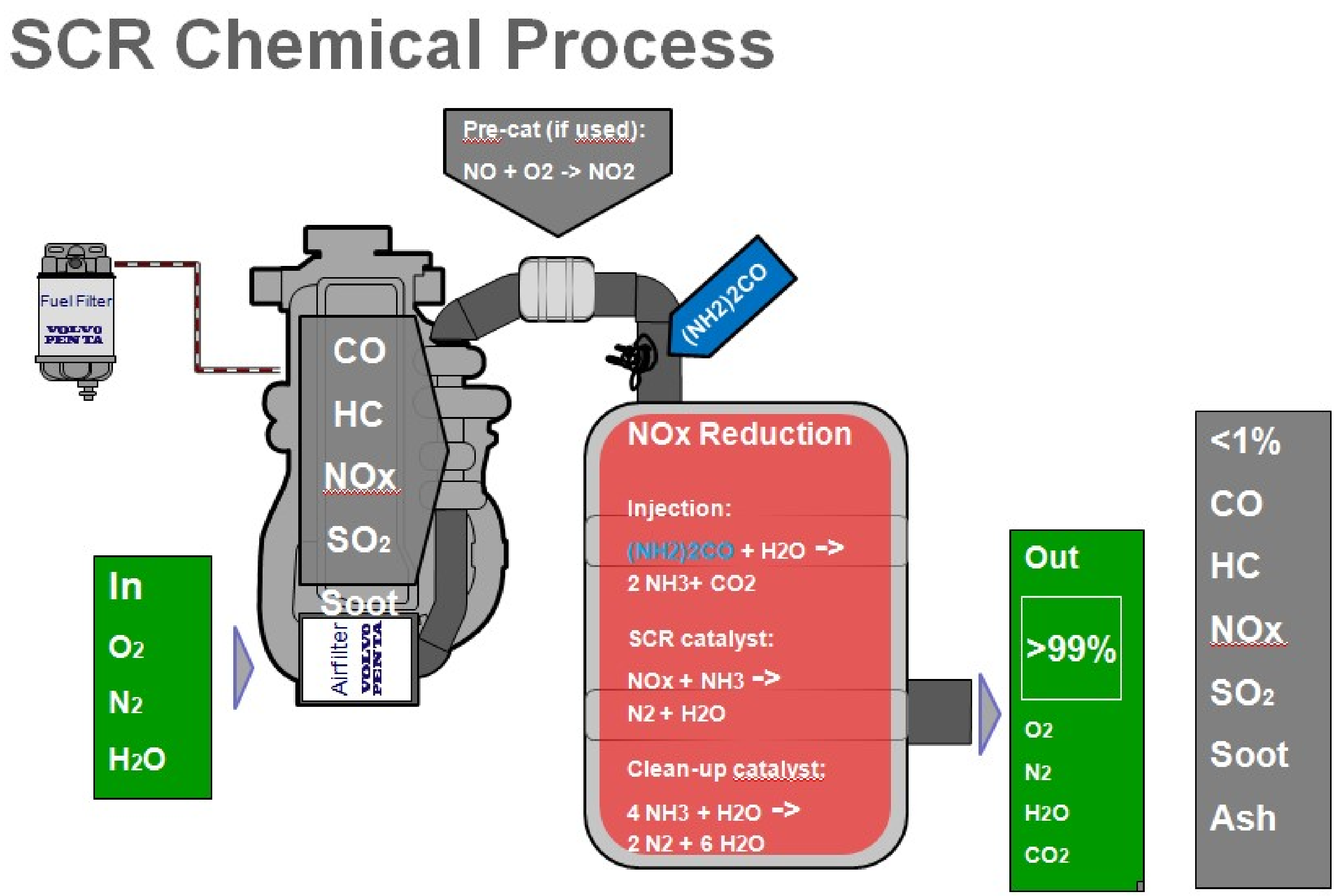


**Figure 4.4: SCR system chemistry and processes (Volvo Penta).**

Field trials have demonstrated that SCR systems can reduce NOx from 60 to 65% and DPM by up to 25% (Rubelli and Cassidy, 2010).

In coal mines, where their use is often mandated (i.e., in the U.S.), diesel equipment may be fitted with disposable exhaust filters that are used in conjunction with heat exchangers or water jackets that are designed to reduce the temperature of the exhaust gases prior to emitting them. This reduction in exhaust temperature allows the use of inexpensive paper filters that are up to 97% efficient at reducing DPM from the engine exhaust. Despite their relatively high efficiencies and inexpensive replacement filters, these systems have not been widely adapted by industries outside of coal mining.

# 4.6 Enclosed Vehicle Cabins

Enclosed cabins on diesel-powered equipment have been proven effective at reducing the exposure of operators. If they are well-sealed and maintained, operator's cabs that are provided with filtered air are 43 - 90% efficient at reducing DPM (Noll and Grau, 2008).

The effectiveness of these cabins is negatively affected any time the doors, windows or seals are damaged. This can be mitigated somewhat by making frequent inspections of the equipment and immediately repairing any deficiencies.

A team of NIOSH scientists identified several factors in the design of vehicle cabs that can impact their ability to protect the operator from dust and DPM and used that knowledge to design a new type of cabin for mining equipment that is capable of increasing the protection factor (equal to the level of dust in the ambient air divided by the dust concentration inside the cab) by an order of magnitude.

A schematic of the vehicle cabin and filtration system components designed by the NIOSH team is shown on Figure 4.5.

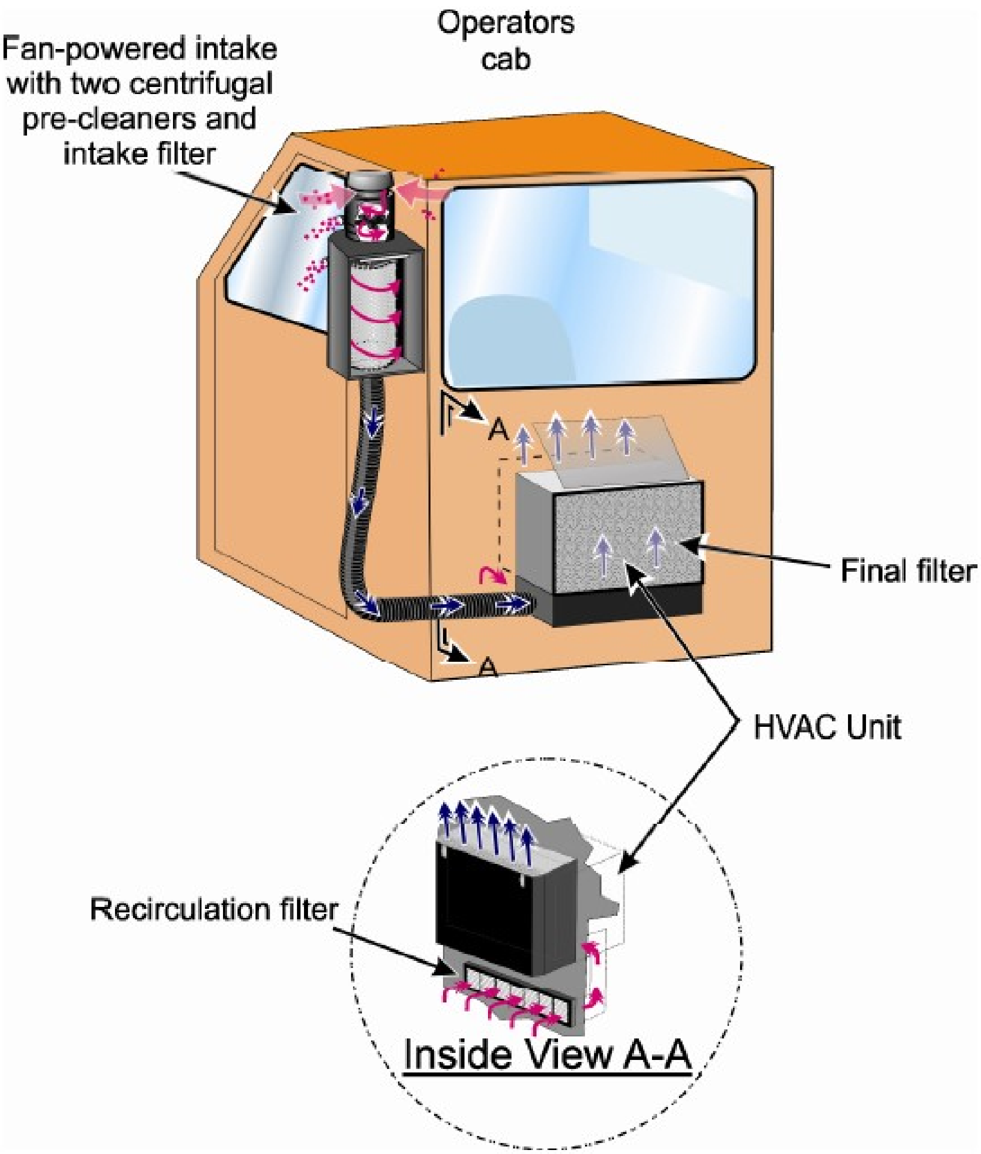


**Figure 4.5: Vehicle cabin pressurization and filtration System (Cecala, 2012).**

Typically, most operator cabins on modern diesel-powered mining equipment incorporates much of the same design elements (on vehicles that include enclosed cabs), and at the very least include effective door and window seals coupled with an intake supply that passes through a hepa filter prior to entering the enclosure.

# 4.7 Administrative Controls and Strategic Initiatives

In addition to the technological solutions, a wide range of administrative controls and strategies have proven to be effective in reducing the diesel emissions exposure of underground personnel.  It should also be noted that not all administrative controls may be allowed in all mines; specifically, rotating workers in jobs or locations with high ambient levels of DPM for the expressed purpose of reducing their exposure is explicitly forbidden in the U.S. (71 Federal Register 28924, 2006).

One commonly used method for reducing the production of diesel emissions in underground mines is to limit engine idle time.  It is easy to implement, requires no equipment modification or training and may actually save money.  Most modern diesel equipment will not be harmed by frequent stops and starts provided that the engine's cooling systems is functioning properly.

Many mines have implemented a vehicle "Tag Board", that limits the number of pieces of equipment (or total horsepower) allowed in an area based upon the ventilation quantity in that area.  This may be done physically, with tags that hang on a wooden board, electronically with RFID markers or by a central dispatcher tracking the location of equipment by radio.  In mines with advanced, ventilation on demand type systems, the ventilation quantity itself may be adjusted based on the equipment present.

Another method for reducing emissions exposure levels for underground workers is to remove them from the environment where the emissions are present.  This is practically accomplished through the use of automated or remotely controlled equipment, particularly in areas where exposures are the highest (e.g., crushers, rockbreakers, grizzly's).  Even if the operator cannot be completely removed from the machine, automating the process sufficiently that it can be accomplished from within the enclosed vehicle cabin can have equal benefits in terms of DPM exposure (e.g., using automated jumbos capable of automatically loading drill-rods).

The last resort in controlling diesel emissions exposure in underground mines should be considered the personal respirator. Although personal respirators have been demonstrated effective, they should not be considered as part of any emissions abatement program unless all other options have been exhausted.  If personal respirators are deemed necessary, they should only be administered as part of a comprehensive plan that includes training, fit-testing, maintenance and replacement when necessary.

# Dust

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Dust exposure has long been known to be a serious threat to workers in many industries both in terms of health and ignitions. In mining, overexposure to respirable coal mine dust can lead to coal workers' pneumoconiosis (CWP). CWP is a lung disease that can be disabling and fatal in its most severe form. In addition, miners can be exposed to high levels of respirable silica dust, which can cause silicosis, another disabling and/or fatal lung disease. Once contracted, there is no cure for CWP or silicosis. Explosion hazards from the presence of suspended volatile particulates are also present in underground mines, especially coal mines. Both primary and secondary explosions resulting from coal dust have caused significant damage and loss of life throughout the history of mining in the U.S. The goal, therefore, is to limit worker exposure to both respirable and explosive dust to prevent these consequences.

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## Course Summary:

| Date | Details |
|---|---|
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# 1.1 Respirable Dust

Exposure to respirable dust has been an occupational hazard for mine workers since the inception of modern mining in the U.S. over 300 years ago. Of course, occupational dust exposure is not limited to mining but is also a significant concern in many other industries such as construction and manufacturing. Although the material presented in this course may be related to these other industries, the presented content will specially address respirable dust in terms of mining. What exactly is respirable dust and what are the issues associated with respirable dust exposure? In essence, respirable dust is simply an airborne particle that is small enough to invade the alveoli of the lungs and trigger the growth of scar tissue. Broad understandings similar to the previous statement are commonly accepted throughout the mining industry. Even when an exposure progresses to the disease stage, a clear distinction between the two conditions, CWP and silicosis, is not available. Frequent use of the simple defining characteristics such as geometric diameters are also used to classify respirable dust. Although accurate, the resulting definitions present an over simplified presentation of respirable dust and its health effects.

Simplistic dust characterizations do not fully explain how dusts behave in its airborne state. A more accurate descriptor for most occupational hygiene situations is “particle aerodynamic diameter.” The aerodynamic size of a particle is defined as the diameter of a hypothetical sphere with a density of 1 g/cm3 that has the same terminal settling velocity in calm air as the hazardous particle in question. The terminal velocity is between the hypothetical sphere and the dust particle(s) is assumed to be equal regardless of its geometric size, shape, and density. This approach to respirable dust classification is more accurate than stating a simple size for describing respirable dust because aerodynamic size is closely related to the ability of a dust particle to penetrate and deposit in the respiratory tract. A conceptual picture of respirable dust entering the respiratory tract is displayed below.

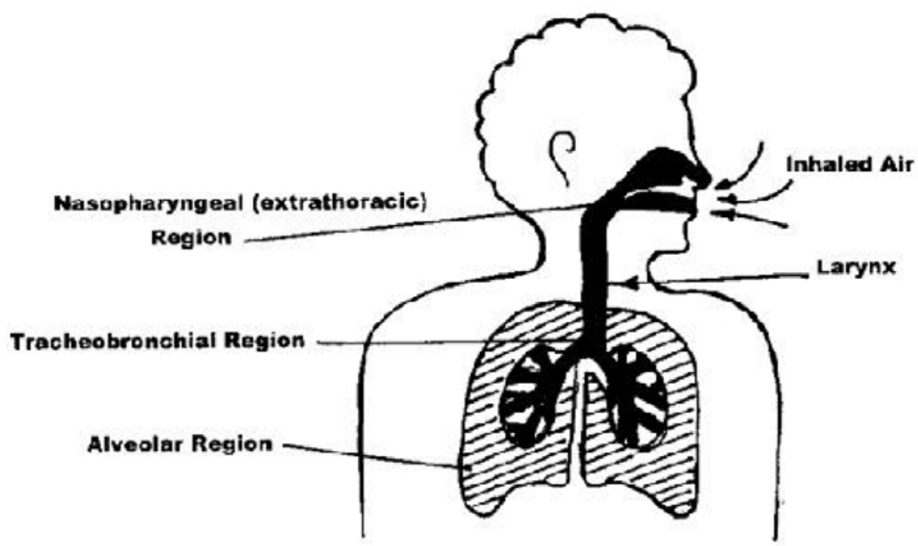


Figure 1.1. A conceptual picture of respirable dust entering the respiratory tract

This characteristic can also be utilized to quantify particle transport in dust sampling and filtration devices. In aerosol science, particles with aerodynamic diameters >50 μm generally do not remain airborne very long and thus do not pose a significant threat unless exposure occurs at the source. However, depending on air flow conditions, particles even >100 μm may become suspended for extended periods of time. Respirable dusts have an aerodynamic diameter of <10 μm. In the upper size range (i.e., close to 10 μm), approximately 1% of these relatively larger respirable particles are able to circumvent normal respiratory filters and deposit in the alveoli. Lung alveoli deposition is maximized when respirable dust particles have an aerodynamic diameter of 2 μm.

Although more accurate, the complexity of dust behavior both in the air and inside the lungs may make other properties appropriate for characterization depending on conditions. For example, dust composition may be useful for determining the exact impact of the dust or diffusion characteristics. This information can then be used to predict how long respirable dust will remain suspended under a set of conditions. As can be seen by this brief overview, respirable dust is a complex subject that incorporates many disciplines from physics to medicine. Further refinement of these descriptors is essential for determining the exact behavior of respirable dusts so that effective controls can be developed to improve mine worker health and safety.

Exposure to respirable dust increases a worker's risk for developing health complications. However, exposure does not result in an immediate negative heath effect. The development of a significant dust-related disability results from long-term exposure and can have a drastic impact on daily activities. Inhaled respirable dust particles that are able to be deposited into the alveolar regions are engulfed by macrophage cells, phagocytes. Under normal circumstances, the macrophages carry the insoluble particle to the ciliated epithelium for removal from the respiratory system. However, once engulfed by the phagocytes, respirable dusts can also remain in the pulmonary space or be carried to the lymphatic system. Coal dust and silica-containing dusts are cytotoxic and can kill the macrophage cells thus causing the formation of scar tissue. The point at which the alveolar cleansing system becomes overwhelmed thereby allow the growth of significant scar tissue marks the transition into the two diseases associated with dust overexposure.

The two general disease classifications of these disease are coal workers' pneumoconiosis (CWP) and silicosis, both of which describe degenerative conditions that affect lung function, such as breathing. Essentially, CWP results from respirable coal dust exposure and silicosis results from respirable silica dust exposure though the latter designation is commonly used to describe the disease from all other types of dust exposure. Other dusts in addition to silica include cement; lead, cadmium, nickel, and beryllium. Despite the trivial differences in the names, both resulting lung diseases can be disabling and fatal in their most severe form. No known cure for CWP or silicosis is available once contracted. The primary objective for mine operators and worker is to limit exposure to respirable dust though the use of good working practices and engineering controls.

# 1.1 Explosive Dust

Explosive dust is an inherent hazard for underground mines with many recent mine event linked to catastrophic secondary ignitions of mine dusts. A dust explosion can cause destruction of infrastructure, injury to personnel, and loss of life. Despite the seemingly mine-specific nature of explosive dust, a sentiment created by the ubiquitous use of the term “coal dust explosions” in the media, this hazard is actually present in numerous industries. Combustible dusts can essentially be composed of any fine particulate that present an explosion hazard when dispersed in air under certain conditions. As a result, explosive dust hazards are found in industries such as agriculture, manufacturing, and chemical refining and can present itself in multiple physical forms. For example, egg whites, powdered milk, cornstarch, sugar, flour, grain, potato, rice, aluminum, bronze, magnesium, zinc, sulfur, pesticides, and textiles are all potentially explosive when rendered in an aerosolized particulate form. Although the material presented in this course may be related to industries other than mining, the content will only address topics in terms of the mining industry.

Coal dust is the primary constituent for the majority of dust related explosions in the underground mining industry. In order for coal dust to become an explosive hazard, it must fulfill a set of basic criteria. These criteria can be generally represented as follows:

1. The dust must be suspended in the air.
2. The dust must be of a particle size capable of sustaining a cascade ignition.
3. The concentration of the coal dust suspension must be within its explosive range for a given set of environmental conditions.
4. An ignition source must be present within or around the dust suspension.
5. The atmosphere must contain sufficient oxygen to support and sustain combustion.
6. A form of geometric confinement is present that allows pressure to accumulate within the dust suspension.

Factors that influence dust explosion severity include moisture content, ambient humidity, and particle shape. Unfortunately, coal dust can achieve the aforementioned criteria more easily and also has a broader explosive rage relative to other industrial dusts. This property combined with the continuous agitation of coal dust by ventilation flow and mining equipment create a significant hazard for dust explosions in underground mines.

The risk of a coal dust explosion increases with decreasing particle size and increasing coal volatility. Despite the omnipresent nature of coal dust in underground coal mines, ignitions in which the primary explosion resulted from coal dust are rare because of modern preventative practices. Techniques applied in the underground mining industry for both preventing and mitigating dust explosions will be discussed in another topic under this course. However, even secondary coal dust explosions precipitated by the primary ignition of a gas can result in severe consequences.

This propensity for damage and injury results from how a coal dust, or any other industrial dust, explosion propagates. When a dust explodes, it deflagrates in the air. In a deflagration, the ignited substance releases heat, hot gases, and energetic particles that ignite and spread the energy. During a dust explosion, the deflagration processes occurs very rapidly. This swift propagation causes the heated air and gaseous fire products to produce extreme air pressure that produce a concussion wave throughout mine infrastructure. Given the confined geometry of underground coal mines, the concussive force is further amplified thus potentially resulting in even greater damage. The explosive wave can additionally suspend otherwise settled dust on the floor, walls, and roof and provide additional fuel for a tertiary dust explosion. The primary objective for mine operators and workers to prevent dust explosions are safe working practices and engineering controls.

# 2.0 Introduction to Controls

Dust is an inevitable byproduct of a mining operation whether located underground or on the surface. This material as a static substance, such as settled on a piece of equipment, is fairly benign and only creates a visual nuisance. However, dust can quickly become harmful to mine personnel when it is aerosolized in particulate form. Unfortunately, the complete prevention of suspended dust is impractical, but a variety of engineering controls designed to reduce dusts hazards are available. The following discussion provides a general overview of personal and engineering controls commonly utilized for reducing worker exposure to respirable and explosive dust.

# 2.1 Personal Controls

Dust is an inevitable byproduct of mining whether located underground or on the surface. Unfortunately, suppression controls cannot completely remove the health hazards posed by respirable dust. A certain amount of personal precautions must be taken by workers in high dust areas to reduce these hazards. The following discussion provides a general overview of personal protective equipment (PPE) and practices for reducing worker exposure to respirable dust.

Respirable dust is created during mining activities that involve rapid physical agitation such as mechanical extraction, ventilation, equipment movement, ore transport, etc. At these production locations, PPE designed to filter respirable particulate is generally either required or highly recommended by mine operators. Two main types of PPE are applied for this purpose, powered air purifying respirators and passive breathing masks. The powered respirator is composed a face shield, helmet, fan, filter, and battery pack. Essentially, air is actively pumped through an air filter to remove all respirable particulate. The clean air is then pressurized across the face of the user to create a positive pressure area of filtered air directly behind the face mask that exits from the bottom. The pressurized air not only provides a source of clean air for the user, but prevents fugitive dust from entering the mask. Maintenance of powered respirators is straightforward and simply requires cleaning, filter replacements, battery recharging and replacements, and helmet replacements every 3 years.

Passive breathing masks are available as disposable face masks or as manual respirators with replaceable filters. This type of personal breathing filter is the simplest and least expensive of the two respirator types available. Manual respirators are designed to prevent particulates from entering the lungs by capturing them in a filter rated for the mining environment in which they will be applied. The passive nature of these breathing filters requires more effort from the user relative to the powered respirators. However, the overall footprint is much smaller because no powered systems are needed. Additionally, passive respirators require that the area of the face where the respirator seal will lie be free of particulates and hair to ensure a tight seal. As a result, donning such a respirator may feel inconvenient to certain workers and discourage their use. If utilized and maintained properly, passive respirators are extremely effective in preventing the inhalation of respirable dust. Workers may also choose to adopt some simple routine practices to limit their exposure.

Workers can recognize when and where respirable dust may be generated and plan ahead to avoid or mitigate the source. For example, if rock dusting is scheduled in a certain location, workers may choose a different travel route to avoid the area. If avoidance is not practical or impossible, the worker can utilize an available breathing filter supplied by the mine. In a scenario where a worker may generate respirable dust in an area void of engineering controls, he or she may request a water spray system in addition to acquiring proper breathing filters suited for the desired situation. Awareness and planning are essential to the prevention of respirable dust related health issues. The feelings of inconvenience and irritation frequently prevent personnel from implementing protective practices for respirable dust. In order to combat this issue, mine operators can institute training programs or incentives to workers who follow and practice recommended respirable dust safety.

# 2.2 Engineering Controls

**Respirable Dust**

Respirable dust is created during mining activities that involve rapid physical agitation such as mechanical extraction, ventilation, equipment movement, ore transport, etc. As such, many of the engineering controls are located at these points of agitation in an effort to reduce the amount of respirable dust. One common control applied for this purpose are water sprays. Water essentially entraps dust particles into conglomerates, which prevents them from entering the respiratory system in a harmful manner. Water sprays are usually used in two main applications, to reduce the production of suspended dust particles due to agitation and to suppress already aerosolized dust particles.

The creation of respirable dust can be reduced by wetting surfaces subject to physical agitation, such as equipment travelways, active mining faces, and ore contained on moving conveyors. Water application to these areas of the mine can help control respirable dust but only if the moisture content is maintained. Several factors can affect how quickly moisture escapes from wetted surfaces, which will in turn affect the frequency of water application. Such factors include the amount of moving air to which the surface is exposed, the seasons (e.g., lower ambient humidity during winter months), and the presence of hydroscopic compounds, such as calcium, magnesium chloride, hydrated lime, and sodium silicate. In contrast to the first two factors, the presence of hydroscopic compounds can improve the retention of moisture thus requiring less frequent water application. Water sprays, as previously introduced, can also be used to suppress dust particles that have already become suspended in the air.

Water spays designed to suppress aerosolized dust are typically located on mining equipment used for ore extraction, such as continuous miners, longwall shearers, and road headers. These sprays direct water toward the point of extraction so that any generated dust can be quickly entrapped and brought harmlessly to the floor. The effectiveness of this control is indirectly proportional to the size of the applied water droplets. The smaller the water droplet size, the greater the wet surface area, and the more dust that can be suppressed. As a result, water sprays designed to suppress aerosolized dust produce a pressurized mist as opposed to streams of water. However, if not properly applied, water sprays can actually increase airborne respirable dust levels under certain circumstances. An example of machine mounted water sprays is provided below.

Figure 2.1. Water sprays mounted on a longwall shearer.

Sprays that deliver water at too high of a pressure for a given application can force liberated dust from the extraction surface and into the mine ventilation airflow (Jankowski and Colinet 2000). In order to balance these two traits, a variety of water spray nozzle designs are available. Currently, full-cone sprays appear to be the most effective type of spray pattern providing a balance between surface area, water quantity, and delivery pressure. Additional research is being conducted to determine the most effective spray patterns and mounting locations as a function of application (i.e., continuous miner, longwall shearer, etc.). Additives to the water may also be useful in enhancing the dust suppression capability of water sprays. Examples of different spray nozzle designs are provided below.

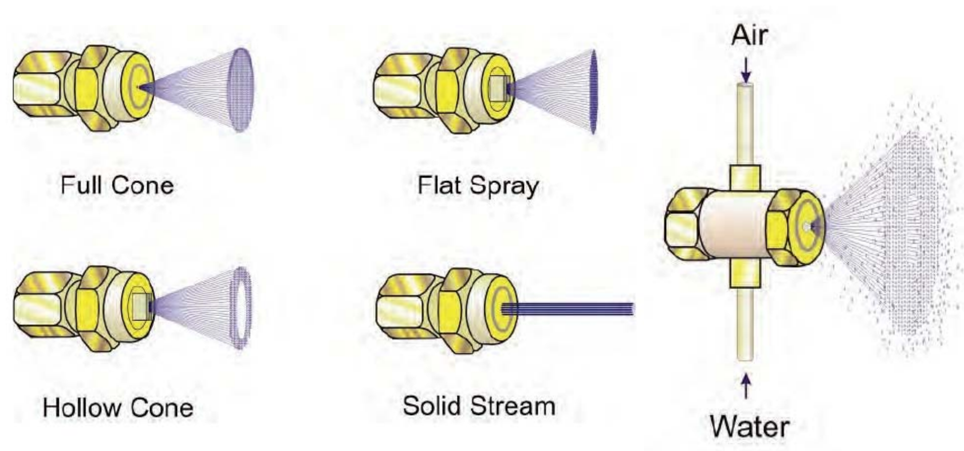


Figure 2.2. Nozzle spray designs

Wet dust suppression fundamentally requires the formation of microscopic liquid films as a means of increasing the adhesion of dust particles. Additives that enhance this ability, such as surfactants, may also improve the ability of water sprays for reducing dust production from static surfaces as well as for suppressing already suspended dust. Although water additives for preventing dust suspension are not commonly utilized in mines, they are frequently applied in other industries such as road construction and agriculture. For example, suppressants used to control dust on unpaved roads in agriculture include brine solutions created using sodium chloride (NaCl), calcium chloride ($CaCl2$), and magnesium chloride ($MgCl2$).

While lignin, asphalt emulsions, natural clays, plant oils, and chloride solutions are the predominant dust control additive in many industries. These additives have the additional advantage of being hygroscopic in nature, a category of material that pulls moisture from the air. Applied by itself, water is subject to evaporation, which can be extreme in certain situations. In these cases, water must be reapplied frequently to maintain adequate moisture, which translates to high operating costs. The addition of a hydroscopic material provides a means to mitigate this issue.

A variety of surfactants have been added to water sprays designed to suppress suspended dust at ore extraction points. These additives have demonstrated inconsistent levels of success in such applications because of the variability in mining conditions from site to site. However, research evaluating sprayer design and position in conjunction with additives continues in an effort to determine the usefulness of surfactants for dust suppression. Water sprays, although effective when applied properly, are limited to the area in which they are applied. Other engineering controls and practices are utilized to supplement water sprays in other mine locations.

Ore transfer points along equipment haulage and conveyor routes are locations subject to respirable dust production. Fully enclosing these transfer points, such as at the stageloader, crusher, feeder, hopper, etc., is an effective method for reducing dust production. Common practices include installing a combination of steel plates, strips of conveyor belting, or other such materials to enclose the transfer point. This enclosure encompasses the entire transfer point as well as along the conveyor after the transfer point subject to dust dispersion. Proper maintenance of equipment, especially conveyor systems and water sprays, is essential for controlling respirable dust. The proper function of dust suppression equipment must be maintained to ensure that mine workers are not exposed to an unexpected dust levels. Properly maintaining the conveyor system not only benefits production but simultaneously prevents the production of aerosolized dust in unmitigated areas. For example, missing rollers, belt slippage, and worn belts can create significant spillage zones that push dust into the ventilation stream (Organiscak et al. 1986).

As with all underground mining methods, ventilation is used to both bring fresh air to working areas and to dilute hazardous gases, such as methane, to safe levels. Ventilation also has a secondary effect of controlling respirable dust in active mining areas. As with hazardous gases, ventilation air dilutes and carries aerosolized dust from these sections and into the return areas of the ventilation system. However, ventilation also provides a source of agitation for any dust that escapes primary controls. Depending on the mining conditions present at a site, a certain range of velocities may serve to produce respirable dust to a degree that exceeds its diluting capabilities. In this scenario, additional controls can be implemented to mitigate the fugitive dust created by the ventilation system.

**Explosive Dust**

Explosive concentrations of dust employ many of the controls implemented to suppress respirable dust with the added practice of rock dusting. Underground mining can produce a variety of fine particulate that can serves as a source for dust explosions under certain conditions. In situations where accumulations of explosive dusts become hazardous, limestone powder, known as rock dust, is deposited throughout the mine workings on a regular basis. This practice is required in all underground coal mines to prevent coal dust explosions. Rock dusting prevents dust explosions by providing a sufficient amount of inert material to dilute explosive concentrations of hazardous dusts. However, the proper proportion of rock dust to explosive dust must be achieved to reduce the overall concentration below the explosive threshold.

In the event of an explosion, rock dust also helps to reduce the chances for a catastrophic chain reaction. This mitigating effect is created by the dispersion of the limestone powder when the rock dust contacts the explosive pressure wave. The suspended limestone particulate is then able to absorb the heat generated from the explosion, which either stops the chain reaction or reduces the intensity of the propagating deflagration. Rock dusting, although effective, is a control that must be continuously applied. Even thin layers of explosive dust settled on previously rock dusted areas can create an explosive condition. Other preventative strategies for explosive dust hazards include dust inspection, testing, housekeeping, and control programs. These initiatives would supplement in-place rock dusting policies by minimizing the accumulation of explosive dusts, checking for the proper proportion of rock dust for a given explosive dust, and ensuring that dust control equipment is properly maintained.

# 3.0 Introduction to Regulation

The following module provides a summary of the most current U.S. Federal regulations that pertain to dust requirements for both coal and metal/non-metal mines. Depending on policies enforced by States, localities, and individual mining companies, the requirements introduced in this module may be less stringent than what is required at a given mine site.

# 3.1 Coal Mines

**Respirable Dust Regulations**

Dust regulation has recently been revised to address the concern with rising occurrences of dust related diseases. These new regulations are being implemented on a two-year phase-in schedule, which began in August 2014. An overview of changes, some specific requirements, and implementation schedule are provided as follows.

• Reduces the maximum concentration of coal dust allowed in the air
• Increases sampling requirements
• Requires the use of the continuous personal dust monitor (CPDM) for dust sampling (figure of CPDM provided below)

Figure 3.1. Continuous personal dust monitor

• Requires immediate corrective action when regulatory dust limits are exceeded
• Provides MSHA with the ability to issue citations based upon a single full shift sample that exceeds the citation level
• Dust samples can no longer be averaged
• Mandates immediate action by mine operators when dust levels are high
• Requires more frequent sampling of areas known to have high dust levels, such as those closest to the production area
• Requires sampling for the full shift a miner works
• Requires dust samples to be taken when mines are operating at least 80% of production, as opposed to the existing 50% requirement
• Requires more thorough examinations of the dust controls on mining sections each shift with records of the exams signed by mine officials
• Lowers the overall dust standards in underground and surface coal mines from 2.0 to 1.5 milligrams per cubic meter of air (mg/m$^3$) in areas of the mine where coal is produced
• Halves the existing standard from 1.0 to 0.5 mg/m3 in mine entries used to ventilate areas where miners work and for miners who have reported evidence of pneumoconiosis
• Certified persons who perform dust sampling and who maintain and calibrate sampling equipment must:
Complete an MSHA course of instruction
Pass an MSHA examination (Sample Exam Questions: **Certified Person Sampling CPDM Exam Sample Questions.pdf (https://canvas.instructure.com/courses/1049387/files/46381386/download?wrap=1)** **(https://canvas.instructure.com/courses/1049387/files/46381386/download?wrap=1)** **(https://canvas.instructure.com/courses/1049387/files/46381386/download?wrap=1)** )
Renew certification with MSHA every three years
• MSHA allowed to revoke a person’s certification for failing to properly carry out the required sampling or maintenance and calibration
• Makes Lung function testing available to all coal miners
• Expands chest x-rays to include surface miners
• Expands dust sampling for working miners with evidence of a dust related disease

**Implementation Schedule**

August 1, 2014: The following provisions were effective immediately on this date:
• Operators required to take immediate corrective actions to lower respirable dust concentrations when any operator-collected sample exceeds the regulatory concentration limit
• Method of averaging samples changed
• Sampling required for the full shift that a miner works
• Samples required to be collected when the production on the mining unit is near normal levels, at least 80% as opposed to 50%
• Ventilation plans upgraded to specify the individual dust controls used on each mechanized mining unit
• More thorough and verified exams required of the dust controls each shift
• Noncompliance determinations based on a single full-shift MSHA-collected respirable dust sample
• Sampling requirements increased for surface mines
• Sampling required on all shifts
• Sampling frequency increased for miners with evidence of a dust related disease
• Medical surveillance program expanded to include lung function testing, occupational history, and symptom assessment as well as x-rays to all coal miners
• Surface miners with evidence of a dust related disease given the right to transfer to a work assignment in a less dusty area of the mine

February 1, 2016: Mine operators are required to implement the CPDMs sampling technology.

August 1, 2016: Mine operators are required to achieve the final 1.5 and 0.5 mg/m$^3$ standards. In addition, the 1.0 mg/m$^3$ standards for miners with evidence of the disease and for intake air ventilating the mine are lowered to 0.5 mg/m$^3$

**Explosive Dust Regulations**

The U.S. regulatory requirements for rock dusting are stated in 30 CFR 75, Subpart E (Combustible Materials and Rock Dusting). This regulation requires the use of rock dust in bituminous coal mines (30 CFR 75.402) to abate the hazard of accumulated coal dust. Rock dust must be distributed upon the top, floor, and sides of all underground areas. The amount of applied rock dust must be sufficient to achieve a percent of incombustible content of coal dust, rock dust, and other dust greater than 65%. All areas of a coal mine that can be safely traveled must also be kept adequately rock dusted to within 40 ft of all working faces. In the return area of a coal mine, the proportion of incombustible material must be greater than 80% (30 CFR 75.403).

Rock dust is officially defined as follows by MSHA:

Pulverized limestone, dolomite, gypsum, anhydrite, shale, adobe, or other inert material, preferably light colored, 100% of which will pass through a sieve having 20 meshes per linear inch and 70% or more of which will pass through a sieve having 200 meshes per linear inch; the particles of which when wetted and dried will not cohere to form a cake which will not be dispersed into separate particles by a light blast of air; and which does not contain more than 5% combustible matter or more than a total of 4% free and combined silica ($SiO_2$), or, where the Secretary finds that such silica concentrations

# 3.2 Underground Metal/Non-Metal Mines

The U.S. regulatory requirements for underground metal/non-metal mines regarding dust are stated in 30 CFR 57, Subpart D. These requirements are must less stringent relative to coal mines. Regulations explicitly stated in 30 CFR 57 cover only asbestos dust exposure. For all other contaminants, MSHA requires that the exposure to these substances not exceed, based on a time weighted average, the threshold limit values (TLV) adopted by the American Conference of Governmental Industrial Hygienists, as set forth and explained in the 1973 edition of the Conference's publication, entitled “TLV's Threshold Limit Values for Chemical Substances in Workroom Air Adopted by ACGIH for 1973,” Pg. 1-54. Excursions above the listed thresholds must not exceed the limits defined as permissible by this document. Sampling is required to be conducted as frequently as necessary to determine the adequacy of control measures. Employees must be withdrawn from areas where there is an airborne contaminant that exceeds its TLV.

MSHA defines asbestos as any material composed of asbestiform hydrated silicates that, when crushed or processed, separate into flexible fibers made up of fibrils. Such materials include chrysotile, cummingtonite-grunerite asbestos (amosite), crocidolite, anthophylite asbestos, tremolite asbestos, and actinolite asbestos. MSHA defines an asbestos fiber to be any particle longer than 5 μm with a length to diameter ratio of at least 3:1. The Permissible Exposure Limits (PEL) states that asbestos exposure cannot exceed an 8-hour time weighted full shift average airborne concentration of 0.1 fiber per cubic centimeter of air (f/cc). No miner can be exposed to airborne concentrations of asbestos in excess of 1 fiber per cubic centimeter of air (f/cc) as averaged over a sampling period of 30 minutes at any time. Sampling is required to be conducted as frequently as necessary to determine the adequacy of controls and exposure compliance.

MSHA requires that asbestos fiber concentrations be determined by phase contrast microscopy (PCM) using the OSHA Reference Method in OSHA's asbestos standard found in 29 CFR 1910.1001, Appendix A, or from an equivalent method. When asbestos monitoring results indicate an exposure that exceeds 0.1 f/cc full-shift limit or the 1 f/cc excursion limit, these samples must be further analyzed using transmission electron microscopy according to NIOSH Method 7402 or by an equivalent method. Employees must be withdrawn from areas where there is an airborne contaminant that exceeds its TLV.

# Fans

Jump to Today  Edit

The fan is the heart of a mine ventilation system but remains one of the more mysterious components of an underground mine. Moving away from the obvious concept such as fans create air flow, the average mining engineer may not be well versed in the operational and functional intricacies of mine fans despite their importance. The following course seeks to mitigate this issue by providing an overview of mine fans including discussions of fan theory, fan types, and surveys. The modules progress from abstract to concrete by first developing a theoretical basis regarding mine fan function and then concluding with operational practices for mine fan. The information presented in this module is compiled from a variety of sources including discussions with ventilation engineers, "An Engineer's Handbook on Fans and their Applications" edited by Robert Jorgensen, "Subsurface Ventilation Engineering" by Malcolm J. McPherson, and NASA.gov. The goal of this course is to provide users with a cursory understanding of mine fans so that this knowledge may be effectively implemented in a given application.

Get Started

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About Aeolus

## Course Summary:

| Date | Details |
|---|---|
| | Module Quiz (https://canvas.instructure.com/courses/1049400/assignments/5495982) |

# 1.0 Introduction

The performance of mine fans is generally represented by a series of curves representing the characteristic variables of pressure, efficiency, and power as a function of airflow quantity. These curves are frequently organized by rotational speed, air density, and fan dimensions. Although useful, curves supplied by fan manufacturers do not represent the entire operational spectrum of a fan. Publishing characteristic curves for all possible speeds and densities would simply not be practical. However, the fan laws provide a means to extrapolate existing curves to other performance points not represented by the available documentation. These transformations can occur because the fan laws represent the proportional relationships between fan speed, flow quantity, pressure, and power. If sufficient existing data is available, the fan laws can also be used to predict fan performance as well as to scale fan designs. Each of the aforementioned capabilities enabled through the utilization of the fans laws are overviewed in this module.

# 1.1 Derivation of Fan Laws

## Euler's Equation for the Conservation of Momentum

Before moving into the derivation of the fan affinity laws, a brief overview of Euler's equation for the conservation of momentum, which is the basis for deriving the fans laws, is useful. The following derivation of Euler's one dimensional momentum equation does not apply only to the fan laws but is rather a general overview of how Newton's Second Law can be used to reveal a number of useful relationships in ventilation. If only the derivation of the fan affinity laws is of interest, this process is provided immediately following the overview of Euler's equation.

The Euler equation for steady flow of an ideal fluid along a streamline represents the relation between the velocity, pressure, and density of a moving fluid. This equation is directly sourced from Newton's Second Law of Motion. An interesting note about Euler's equation is that the integration of the Euler's equations for fluid flow gives Bernoulli's equations.

The following assumptions are made for deriving Euler's equation for the conservation of momentum.

1. The fluid is non-viscous (i.e., the frictional losses are insignificant).
2. The fluid is homogeneous and incompressible (i.e., mass density of the fluid is constant).
3. The flow is continuous and steady.
4. The velocity of the flow is uniform over the section.
5. No energy or force (except gravity and pressure forces) is involved in the flow.

Figure 1 displays a flow region with area, A, though which airflow enters from the left inlet at $x_0$ with a velocity of $u_0$ and exits from the right outlet at $x_1$ with a velocity of $u_1$. Airflow only moves in one dimension from $x_0$ to $x_1$. No flow is present in the y-direction. Air enters the region at $x_0$ with some inherent pressure of $p_0$ and exits the regions with a different pressure of $p_1$.

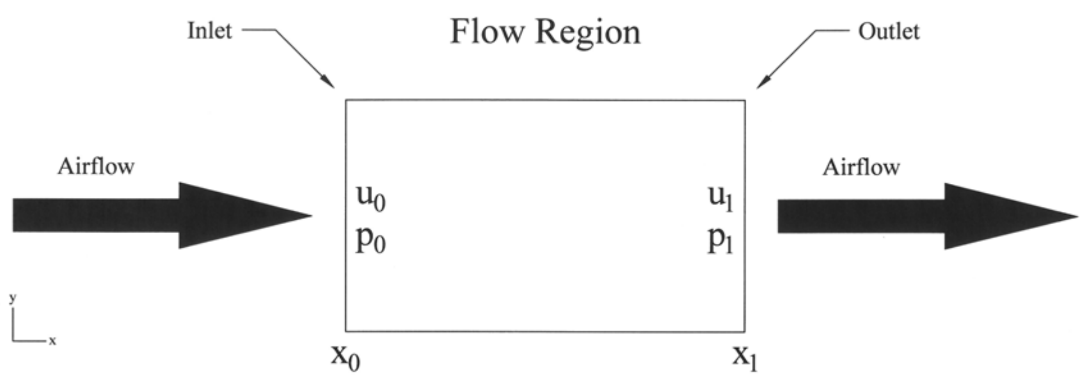



Figure 1.1. Example flow region used to derive Euler's 1D equation representing the conservation of momentum.

The flow is assumed to be incompressible thus making the fluid density constant throughout the flow region. The distance between points $x_1$ and $x_2$ is defined as the change in x or $\Delta$x. The velocity gradient of the airflow is defined as the change in velocity per unit displacement in the x-direction or $\Delta u/\Delta x$. Based on these parameters, the flow velocity at the outlet can be determine by adding change in velocity as the air moves across the defined region from inlet to outlet. The velocity at the outlet can therefore be represented by the following equation.

$$u_1 = u_0 + \frac{\Delta u}{\Delta x} \cdot \Delta x$$

Based on the same conceptual principle, the pressure at the outlet can be represented by the following equation.

$$p_1 = p_0 + \frac{\Delta p}{\Delta x} \cdot \Delta x$$

Newton's Second Law of Motion states that the acceleration of an object is dependent upon the net force acting upon the object and the mass of that object. Following the same concept, the acceleration of an object then depends directly upon the net force acting upon the object and inversely upon the mass of the object. For an object with constant mass, Newton’s Second Law can be represented as F = ma. Since the flow domain is assumed to be incompressible, the mass of air in the region can be represented as follows.

$$F = m \cdot a = m \cdot \frac{\Delta u}{\Delta t}$$

For airflow, force in the system is provided by the pressure gradient across the given flow region. The different in pressure across the region acts on the air to induce flow from the area of high pressure at the inlet to the area of low pressure at the outlet. Since pressure is a force per unit area, the net force on the flow region is the difference between the total pressure at the outlet and the total pressure at the inlet, which gives the following equation.

$$F = -\left(p_1 \cdot A - p_0 \cdot A\right) = m \cdot \frac{u_1 - u_0}{\Delta t}$$

The minus sign at the beginning of the equation appears because the pressure at the outlet is less than the pressure at the inlet. Velocity and pressure can then be represented in terms of the inlet properties to produce the following equations.

$$-\left[\left(p_0 + \frac{\Delta p}{\Delta x} \cdot \Delta x\right) A - p_0 \cdot A\right] = m \cdot \frac{\left(u_0 + \frac{\Delta u}{\Delta x} \cdot \Delta x\right) - u_0}{\Delta t}$$

Removing like terms and simplifying produces the following equation.

$$-\frac{\Delta p}{\Delta x} \cdot \Delta x \cdot A = m \cdot \frac{\Delta u}{\Delta x} \cdot \frac{\Delta x}{\Delta t}$$

Note that $\Delta x/\Delta t$ represents the average velocity across the flow region, u, and that the mass of the region is the air density, ρ, times the volume, V. Volume can be calculated by multiplying the cross-sectional area, A, with $\Delta x$.

$$-\frac{\Delta p}{\Delta x} \cdot \Delta x \cdot A = \left(\rho \cdot A \cdot \Delta x\right) \frac{\Delta u}{\Delta x} \cdot u$$

Removing like terms and simplifying produces the following equation.

$$-\frac{\Delta p}{\Delta x} = \rho \cdot u \cdot \frac{\Delta u}{\Delta x}$$

The $\Delta p/\Delta x$ and $\Delta u/\Delta x$ terms represent the pressure and velocity gradients across the flow region from $x_0$ to $x_1$. If the region is decreased in size to an infinitesimally small volume, or a differential size, then the gradients become differentials. The resulting equation is the 1D Euler equation representing the conservation of momentum for an incompressible, inviscid fluid (i.e., F = ma for an air flow region in the x-direction).

$$\frac{dp}{dx} = \rho \cdot u \cdot \frac{du}{dx}$$

## Fan Law Derivation

The relationship between pressure, density, and velocity represented in Euler’s Equations create the basis for deriving the fan affinity laws. Before moving into the fan law derivation, Euler’s Equation must first be modified to express these terms for fan pressure as a function of fan velocities. For fans, the pressure term is used to represent the change in total pressure across the fan, which is the increase in pressure across the impeller (i.e., the fan total pressure). The density term, identically to the flow region example introduced previously, is used to represent the air density. The average velocity and velocity gradient terms are replaced with the blade tip velocity and the peripheral component the air velocity leaving the blade tip. The resulting Euler Equation is written as follows where $p_T$ is the fan total pressure, ρ is the air density, $v_B$ is the blade tip velocity, and $v_P$ is the peripheral component of the air velocity leaving the blade tip. A conceptual diagram of blade-tip force and velocity vectors with an exaggerated blade width is provided in Figure 2 for an axial flow fan. The 3D conceptual rendering of this impeller inside a fan housing is also displayed in Figure 3. Peripheral velocity from a centrifugal fan is tangent to the direction of rotation.

$$p_T = \rho \cdot v_B \cdot v_P$$

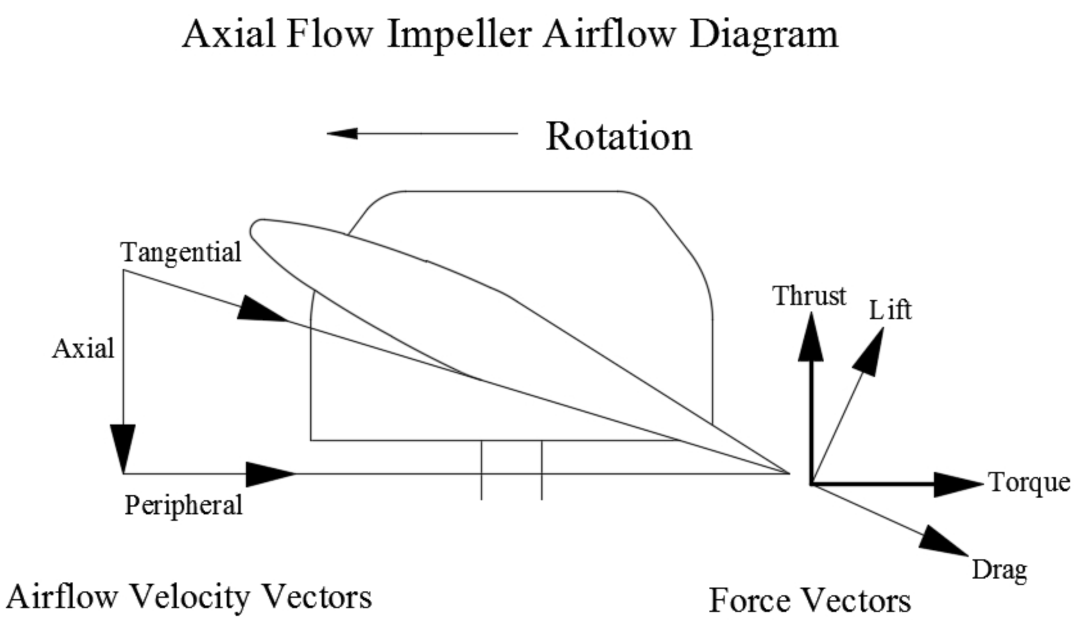



Figure 1.2. Force and velocity vector diagram of an axial flow impeller. The displayed impeller vane is exaggerated in relative scale for clarity but is representative of the 3D conceptual rendering in Figure 3. Given the displayed orientation of the 3D fan, the fan would be rotating clockwise to produce the force and velocity vectors displayed in this figure, Figure 2.

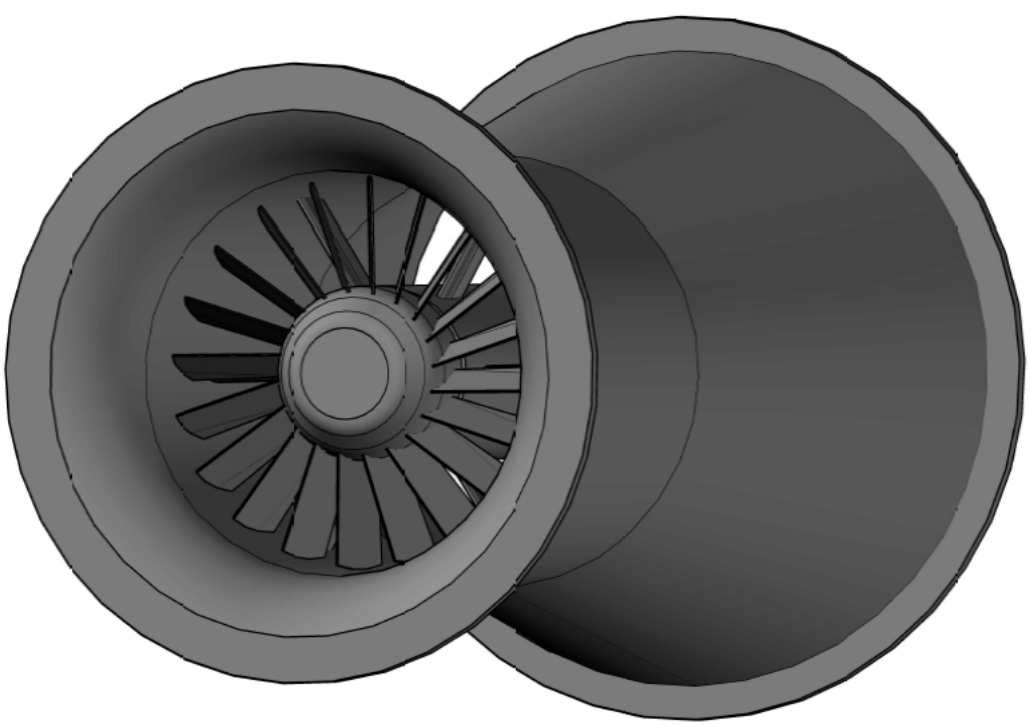


Figure 1.3. 3D Conceptual rendering of axial flow fan with impeller, fan housing, and stationary discharge vanes.

The right hand side of the Euler Equation for fans is derived from the definition of the power consumed by the fan impeller where power is equal to the product of fan total pressure and exhaust quantity (i.e., the rate at which work is done by the fan impeller is equal to the force imparted by the impeller, which is the fan total pressure, to move a quantity of air per unit time). As such, the power consumed by the fan impeller can be written as follows where P is the power, $p_T$ is the fan total pressure, and Q is the quantity of air being moved by the fan.

$$P = p_T \cdot Q$$

The fan total pressure, $p_T$, exerted by the impeller assuming no shock or friction losses is equal to the product of the air density, peripheral blade tip velocity, and peripheral air velocity leaving the blade tip. From this definition, Euler's Equation, or F = ma for a fan, results in the equation $p_T = \rho \cdot v_B \cdot v_P$ presented previously. The peripheral velocities of both the blade tip and the air vary directly with the rotational speed of the fan and the diameter of the impeller blade. As such, fan total pressure is proportional to the product of air density, fan rotational speed, and fan diameter. The following expression can be written to represent this relationship where $p_T$ is the fan total pressure, ρ is the air density, n is the rotational speed and d is the impeller diameter.

$$v_B = v_P = n \cdot d$$

$$p_T \propto \rho\,(n \cdot d)\,(n \cdot d) \rightarrow p_T \propto \rho \cdot n^2 \cdot d^2$$

## Airflow Fan Law

For a centrifugal fan, the radial air quantity at the impeller outlet is equal to the product of the impeller area at the outlet (i.e., circumference of the impeller multiplied by the impeller width) and the radial velocity, $v_R$. To maintain geometric similarity between any two centrifugal fans, impeller width must be proportional to impeller diameter and all vectors must be proportional to each other, which give the following relationships.

$$Q \propto d^2 \cdot v_R$$

$$Q \propto v_B \propto n \cdot d \rightarrow Q \propto n \cdot d^3$$

For an axial flow fan, the air quantity produced at the impeller outlet is equal to the product of the impeller area at the outlet and the axial velocity, $v_A$.

$$Q = \frac{\pi \cdot d^2}{4} \cdot v_A$$

Given that geometric similarity exists between any two axial flow fans, all vectors must be proportional to each other, which produces the same relationship between quantity, tip speed, and impeller diameter previously introduced with centrifugal fans.

$$Q \propto v_A \propto n \cdot d \rightarrow Q \propto n \cdot d^3$$

## Density Fan Law

Based on Euler's Equation for fans, fan pressure varies directly with air density.

$$p_T \propto \rho$$

Note: for fans, volume flow rather than mass flow is typically used for measurements. For example, if air density changes are used for comparing two fans, the final operating point comparison is still made at the corresponding magnitudes of volumetric flow.

## Air Power Fan Law

As previously introduced, the consumed by the impeller to induce airflow can be represented by the following equation.

$$P = p_T \cdot Q$$

Given that $p_T \propto \rho \cdot n^2 \cdot d^2$ and $Q \propto n \cdot d^3$, then power must be proportional to the product of these two relationships.

$$P = \rho \cdot n^3 \cdot d^5$$

# 1.2 Summary and Application of Fan Laws

As previously introduced, the fan laws provide a means to extrapolate existing curves to other performance points as well as to predict fan performance between similar fans. Both of these objectives are completed in the same manner because even the latter case essentially assumes that the fans being compared are identical, a concept that will be explained later. In general, only one of the independent variables, speed, air density, or impeller diameter, are changed thus keeping the remaining two variables constant. In order to predict the performance of a new fan based on the performance of an existing fan, the basic requirement is that the two fans are similar, or homologous, in design and operation. Fans can be identified as homologous when their air passages are geometrically similar and they function at corresponding operating point where the relative shutoff and free delivery locations are the same.

Many variations of the fan laws can be derived to express a number of performance characteristics for a variety of scenarios. These variations are basic mathematical manipulations of the fundamental laws introduced in this module. Regardless of the variation, several properties are always constant between fans. A summary of one interpretation of the fan laws as well as examples of how to apply them are provided in this section. Note that density and compressibility coefficients are always represented as independent variables while velocity pressure and sound power level are always dependent variables. For simplicity, all flow in this module section will be considered incompressible. This assumption allows compressibility coefficients to be removed and simplifying many calculations. A brief overview of compressibility effects will be provided in a later module topic.

**Fan Laws**

| **Speed (n)** | **Impeller Diameter (d)** | **Air Density** $(\rho)$ |
|---|---|---|
| $p \propto n^2$ | $p \propto d^2$ | $p \propto \rho$ |
| $Q \propto n$ | $Q \propto d^3$ | $Q$ Fixed |
| $P \propto n^3$ | $P \propto d^5$ | $P \propto \rho$ |

Based on the relationships shown above, the following mathematical ratios for pressure, quantity, and air power can be used to determine the performance of a new fan, B, to the original, homologous fan, A, or between different operating parameters of the same fan. These ratios can be rearranged in a number of ways to isolate an unknown fan property. In these equations, $p_T$ is the fan total pressure, n is the fan rotational speed, d is the impeller diameter, Q is the airflow quantity, P is the air power, and $\rho$ is the air density.

$$\frac{p_{T,A}}{p_{T,B}} = \frac{n_A^2 \cdot d_A^2 \cdot \rho_A}{n_B^2 \cdot d_B^2 \cdot \rho_B}$$

$$\frac{Q_A}{Q_B} = \frac{n_A \cdot d_A^3}{n_B \cdot d_B^3}$$

$$\frac{P_A}{P_B} = \frac{n_A^3 \cdot d_A^5 \cdot \rho_A}{n_B^3 \cdot d_B^5 \cdot \rho_B}$$

Some example problems are provided below to aid in the understanding of how the fan laws can be applied.

## Fan Law Example 1

***A fan is currently in operation that uses a 36.5 in. diameter impeller to deliver 10,000 ft$^3$/min of air at a fan total pressure of 1.85 in. $H_2O$. The fan impeller is currently set at 600 rpm and consuming 3.4 hp. The air density is 0.075 lbm/ft$^3$. A new fan is being considered for this location. The new fan being considered has a fan diameter of 73.0 in and will operated at 1,200 rpm. At the time of implementation, the average air density is expected to be 0.070 lbm/ft$^3$. Determine the unknown operating properties for the proposed fan emplacement.***

### Solution

For this problem, the known quantities of the new fan are diameter, d, fan speed, n, and air density, $\rho$. The operating quantity, Q, total fan pressure, $p_T$, and air power, P, are unknown for the new installation. The solution to this problem could be approached in a variety of ways. The most straightforward solution path would be to arrange each of the three fan law equations such that the unknown variable is isolated on one side of the equation and then known values are substituted for the variables on the opposite side. This solution path is shown below for fan total pressure, air quantity, and air power where "A" denotes the current fan in operation and "B" represents the homologous fan being considered for installation at the same location.

$$p_{T,B} = \frac{n_B^2 \cdot d_B^2 \cdot \rho_B}{n_A^2 \cdot d_A^2 \cdot \rho_A} \cdot p_{T,A} = \frac{1,200^2 \cdot 73.0^2 \cdot 0.070}{600^2 \cdot 36.5^2 \cdot 0.075} \cdot 1.85$$

$$p_{T,B} = 27.62\, in.\, H_2O$$

$$Q_B = \frac{n_B \cdot d_B^3}{n_A \cdot d_A^3} \cdot Q_A = \frac{1,200 \cdot 73.0^3}{600 \cdot 36.5^3} \cdot 10,000$$

$$Q_B = 160,000\, \frac{ft^3}{min}$$

$$P_B = \frac{n_B^3 \cdot d_B^5 \cdot \rho_B}{n_A^3 \cdot d_A^5 \cdot \rho_A} \cdot P_A = \frac{1,200^3 \cdot 73.0^5 \cdot 0.070}{600^3 \cdot 36.5^5 \cdot 0.075} \cdot 3.4$$

$$P_B = 812.4\, hp$$

Note that unit conversions are not needed when applying the fan laws in this manner as long as the fan properties being compared have consistent units between Fans A and B because a ratio between known operating parameters is being created. As a result, all like units cancel out to create a dimensionless ratio relating Fan A to Fan B, which means that SI units could be mixed with English units as long as the units are consistent between variables (i.e., the speed, diameter, and density all have the same corresponding units). The ratio then produces a transformed quantity with units identical to the original input value. In the case of this example, pressure, quantity, and power were originally presented as in. $H_2O$, ft$^3$/min, and hp, respectively. Thus, the resulting pressure, quantity, and power for the new fan will also be outputted with these units. Of course, if different output units are desired, then a conversion must be done on the solution value.

## Fan Law Example 2

***A fan is currently in operation that delivers 10,000 ft$^3$/min of air at a fan total pressure of 1.85 in. $H_2O$. The fan impeller is***

***currently set at a rotational speed of 600 rpm and is consuming 3.4 hp. The air density is 0.075 lbm/ft$^3$. Based on data from a recent ventilation survey, mine operators would like to increase the air quantity being delivered by the fan to 12,000 ft$^3$/min. At the time of implementation, the average air density is expected to be the 0.075 lbm/ft$^3$. Determine the unknown operating properties for the proposed fan adjustments.***

**Solution**

For this problem, the important aspect to note is that the same fan is under investigation. As a result, Fan A still represents the original fan properties. Fan B is, however, not a new fan but the same fan at a different operating point. Certain static characteristics of a fan, such as the fan diameter, are the same from Fan A to Fan B. Based on the parameters of the problem, the known characteristics of the "new fan" are diameter, d, air quantity, Q, and air density, $\rho$. Fan diameter and air density are constant from Fan A to Fan B, which allows them to be canceled out. The rotational speed, n, total fan pressure, $p_T$, and air power, P, are unknown for the new installation.

The presented solution approaches the problem by arranging each of the three fan law equations such that the unknown variable is isolated on one side of the equation. Constant variables are then canceled out and known quantities are substituted for the remaining variables. The solution is ordered such that that fan law equations with only a single unknown are solved first to provide the required values to solve for the remaining unknown properties. For example, the rotational speed, n, is needed to determine the new fan total pressure, $p_T$. Fan speed can be solved using the air quantity fan law equation and then used to solve for fan total pressure. This solution path is shown below for fan total pressure, air quantity, and air power where "A" denotes the current fan in operation and "B" represents the fan's new operating parameters.

$$n_B = \frac{Q_B \cdot n_A \cdot \cancel{d_A^3}}{Q_A \cdot \cancel{d_B^3}} = \frac{12,000 \cdot 600}{10,000}$$

$$n_B = 720\, rpm$$

$$p_{T,B} = \frac{n_B^2 \cdot \cancel{d_B^2} \cdot \cancel{\rho_B}}{n_A^2 \cdot \cancel{d_A^2} \cdot \cancel{\rho_A}} \cdot p_{T,A} = \frac{720^2}{600^2} \cdot 1.85$$

$$p_{T,B} = 2.66\, in.\, H_2O$$

$$P_B = \frac{n_B^3 \cdot \cancel{d_B^5} \cdot \cancel{\rho_B}}{n_A^3 \cdot \cancel{d_A^5} \cdot \cancel{\rho_A}} \cdot P_A = \frac{720^3}{600^3} \cdot 3.4$$

$$P_B = 5.9\, hp$$

## Fan Law Example 3

***An isolated underground civilization that developed independently from the rest of the world has recently seen a large increase in its population. The now unsustainable growth has forced the expansion of their underground residential network specifically in the 3rd Section of the 10th Sub-quadrant, which is particularly overcrowded. Ventilation engineers have determined that the majority of the ventilation system's in-place components currently has sufficient capacity to serve the additional demand for the short term. However, an increase of 3.28 kst in system flow resistance is expected assuming a transitional factor of 2.1 thereby decreasing overall expected availability by 13 dks. The one question that remains is if the projected demand will exceed the operational tolerances of the primary ventilation fan currently in operation. As a result, the new ventilation fan performance characteristics must be estimated before the final design phase of the expansion project can begin.***

***Currently, the fan delivers a volumetric flow rate of 10,000 cordrick$^3$/keis (cr$^3$/ks) at a pressure of 1.85 mils/siril$^2$ (ml/sr$^2$). The average air density is 0.075 tn/cr$^3$. The impeller is rotating at 600 zad/sr to impart mechanical energy at 3.4 tempests/keis (tp/ks). The proposed ventilation change will require the fan power to increase to 5.0 tp. The air density is expected to decrease slightly to 0.060 tn/cr$^3$ resulting from modifications being made to the environmental controls. What are the expected output quantity, fan pressure, and impeller rotational speed after the proposed changes are applied?***

**Solution**

No typos are present in this example problem. The scenario presented in Example 3 is designed to show that skills and techniques presented in this module can be translated to much broader applications than just the specific mine cases that are introduced. Additionally, this example is designed to covey the usefulness of approaching unfamiliar problems on a regular basis. Many engineering disciplines teach information in a manner that can bias the student toward a particular context of thinking when a more general understanding would be much more useful. In terms of this discussion, a general understanding gives an engineer the capability to isolate the pertinent first principles in a problem and then apply base knowledge to find a solution. Using this approach, engineers can adapt to unfamiliar scenarios in which units or other such properties may be unfamiliar, but the basic function of the system can be related to more familiar, universally applicable principles.

In terms of the mine ventilation engineer, practice problems and application examples are generally introduced in a mining context. At some instance, engineers tend to become ingratiated with the idea of "mining examples" when learning new topics or even reinforcing old ones. For example, during a class when a concept is first introduced, a question of "what does this have to do with mining?" is frequently asked. During activities when general problems are used, many engineers will wonder why the example has "nothing" to do with mining and will come to the conclusion examples that are not "real" and are thus invalid. This type of knowledge bias transforms into confusion or panic when encountered terminology is different or units are not familiar. Example 3 is designed to show that applicable first principles can be applied to an environment that is completely unfamiliar. Although similar issues are rarely as simplistic in nature and may contain truly foreign aspects, knowledge supplementation though research and learning remains an effective approach even in these cases.

The presented solution approaches the problem by first ignoring unnecessary, though seemingly important aspects of the problem. The information is then reduced, isolated, and analyzed for the desired solution. The performance properties of the same fan at a different operating point are being requested, which immediately points to the fan laws. Moving forward with this assumption, what information is available and what information is actually needed to apply the fan laws for this scenario? Since only the fan and the fan laws are in questions, all of the superfluous information dealing with system resistance, location, and background history can be ignored.

As previously mentioned, the important aspect to note is that the question is being asked about the same fan and that the fan laws will be applied. As a result, Fan A still represents the original fan properties and Fan B is the same fan at a different operating point. Certain static characteristics of a fan, such as the fan diameter, are the same from Fan A to Fan B. Based on the parameters of the problem, the known characteristics of the "new fan" are diameter, d, volumetric flow rate, Q, fan pressure, p, air density, ρ, fan rotational speed, n, and fan power, P.

How can these known factors be deduced based on the given values that clearly have no known conversion factor? Key terms such as "volumetric flow rate" and "energy output" can be used as indicators. The base definitions of the first term is a volume per unit time, which refers to quantity in familiar language. For the later term, additional clues from the units are given. For example, the unit "ks" is in the denominator of given rates, which makes this unit signify time. As a result, any value that is a per unit "ks" is actually a per unit time, which can translate to other rates such as power in the previous example as well as velocity, acceleration, etc.

Although the parameters are defined, their units remain unfamiliar and cannot be converted, but does this issue ultimately matter for the defined problem? The answer is"no" because the fan laws relate performance properties between geometrically similar fans by a dimensionless ratio. As a result, as long as the units are consistent, the fan laws can be applied as long as the outputted values are acceptable in the original units. Fan diameter and air density are constant from Fan A to Fan B, which allows them to be canceled out. The unknown variables are output air quantity, Q, rotational speed, n, and total fan pressure, $p_T$.

The presented solution approaches the problem by arranging each of the three fan law equations such that the unknown variables are isolated on one side of the equation. Constant variables are then canceled out and known quantities are substituted for the remaining variables. The solution is ordered such that that fan law equations with only a single unknown are solved first to provide the required value to solve for the remaining unknown properties. For example, the rotational speed, n, is needed to determine the new output quantity. Fan speed can be solved using the air power fan law equation. The resulting rotational speed can then be used to solve for the remaining unknowns. This solution path is shown below for fan rotational speed, air quantity, and fan pressure where the subscript "A" denoting the current fan and "B" representing the new fan operating parameters.

$$n_B = \left( \frac{P_B \cdot n_A^3 \cdot \cancel{d_A^5} \cdot \rho_A}{P_A \cdot \cancel{d_B^5} \cdot \rho_B} \right)^{\frac{1}{3}} = \left( \frac{5.0 \cdot 600^3 \cdot 0.075}{3.4 \cdot 0.060} \right)^{\frac{1}{3}}$$

$$n_B = 735 \, \frac{zad}{sr}$$

$$Q_B = \frac{n_B \cdot \cancel{d_B^3}}{n_A \cdot \cancel{d_A^3}} \cdot Q_A = \frac{735}{600} \cdot 10{,}000$$

$$Q_B = 12{,}250 \, \frac{cr^3}{ks}$$

$$p_{T,B} = \frac{n_B^2 \cdot \cancel{d_B^2} \cdot \rho_B}{n_A^2 \cdot \cancel{d_A^2} \cdot \rho_A} \cdot p_{T,A} = \frac{750^2 \cdot 0.060}{600^2 \cdot 0.075} \cdot 1.85$$

$$p_{T,B} = 2.22 \, \frac{ml}{sr^2}$$

# 2.0 Introduction

Mine fans are categorized as turbomachinery and are generally composed of a rotating impeller usually encased in a stationary housing. In broad terms, the function of a fan is to propel, displace, or move air using the impeller. This definition is fitting since the impeller is the rotating element that transfers energy to the air to induce flow. Impellers are also called wheels, rotors, squirrel cages, propellers, or runners depending on the region, colloquial dialect, and type of fan installation being discussed. For the purposes of this module, the rotating element in a fan will be referred to at the impeller.

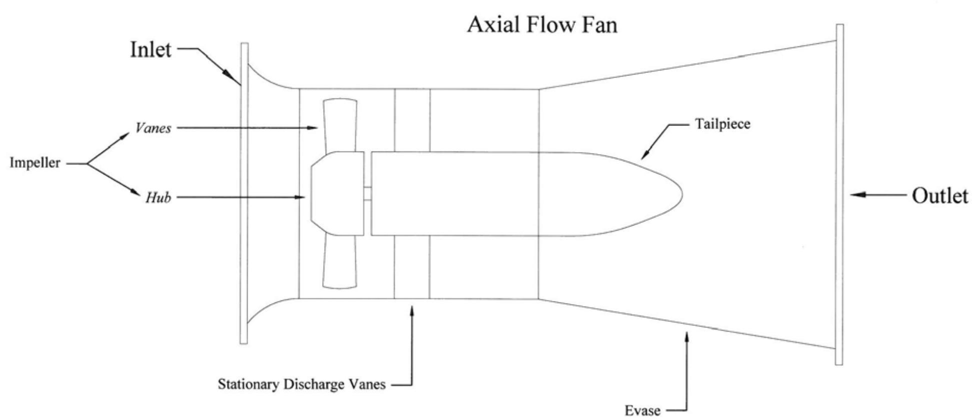
The vanes, which are also frequently referred to as blades, are the principal working surfaces of the impeller. Shrouds and hubs are used to support the vanes and can be composed of a variety of circular forms used to build the connection of the vanes to the center of the fan. A hub may be used to support the vanes directly in which case a shroud is no longer necessary as an intermediary support structure. The housing is the stationary element that guides air into and out of the impeller. Housings are also referred to as casings, stators, and scrolls. The inlet to the fan can have a variety of inlet pieces such as cones, bells, nozzles, or venturi designed to smooth flow by reducing pressure losses.

The outlet from the fan can be equipped with an evase, which is very common in main mine exhausting fans, to reduce losses across the fan by transforming kinetic energy to pressure energy. Stationary vanes may also be installed either upstream or downstream of the impeller to guide air flow into and out of the fan, respectively. Vanes used upstream of the impeller can be called prerotation vanes or inlet-guide vanes and not only smooth flow but can also be angled to efficiently deliver air streams at the optimal angle of attack to the impeller vanes. Vanes used downstream of the impeller can be called straightening vanes or discharge guide vanes. Regardless of the location of the stationary vanes, if any, their basic purpose is to convert pressure energy to the kinetic energy by imparting a tangential velocity to the air, which essentially smooths the air flow to increase fan efficiency. This module provides a general overview of characteristics common to the two major fan types implemented in underground mines, axial flow and centrifugal fans.

# 2.1 Axial Flow

Axial flow fans are one of two major fan types utilized in underground mines. This type of fan is typically applied as a primary mine fan because of its ability to efficiently generate high air flow quantities. However, axial flow fans relative to centrifugal fans operate in lower resistance environments thus increasing the probability of stall during an unexpected event. The nature of the axial flow impeller also makes it vulnerable to substantial damage when operated in stall conditions that result in significant, continuous cavitation of the impeller. Axial flow fans are single-stage machines that induce airflow parallel to the axis of rotation. The impeller is composed of a number of adjustable blades housed in a cylindrical casing that tightly wraps the fan. The clearance between the outer tips of the blades and the inner surface of the casing are minimized to the smallest practical degree. This clearance prevents the created of inefficient impeller-tip vorticies and inefficiencies from losses from axial air flow.

In contrast to centrifugal fans, no velocity component is induced in the radial direction because the flow is confined by the aforementioned casing. As a result, the tangential component of flow generated by the impeller is forced in the axial direction but not completely eliminated. The resulting axial and tangential components of the fluid forces on the rotor produce axial thrust and torque, respectively. These components leave the fan as a whirl of air moving in the direction of the fan's rotation. The discharge vanes, if present, transform the energy from this whirl into useful pressure energy. In order to accomplish this task efficiently, the discharged air is guided along a gradual turn or wing-like surface until the tangential component of the whirl is removed. The discharged flow should still be turbulent but moving effectively uniformly toward the outlet without the presence of radial vortices. If the design of the fan results in an axial velocity that is higher than desired, a diffuser can be used to compensate. A basic conceptual schematic both in 2D and in 3D of an axial flow fan is provided below.



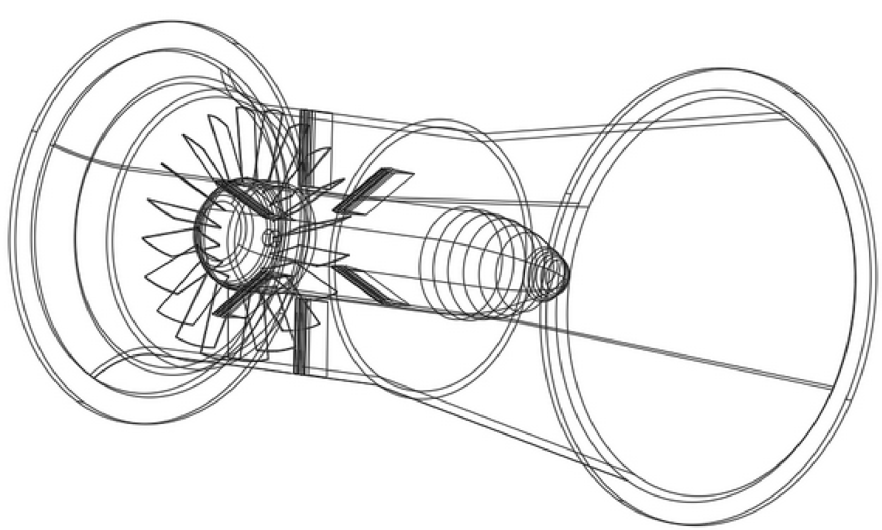



Figure 2.1. Conceptual schematic both in 2D and in 3D of an axial flow fan

The following figure shows the end of an axial flow fan being implemented as a mid-ventilation system booster. Although much smaller in scale and different in design, the booster fan contains the major components of the conceptual schematic shown above.

Figure 2.2. End of an axial flow fan being implemented as a mid-ventilation system booster

The following figure shows the discharge side of an exhausting main mine fan ventilation system equipped with dual axial flow fans. The major external components of the fan housing including the evase and directional ducting can be observed in this picture.

Figure 2.3. Major external components of a fan housing, including the evase and directional ducting

As can be seen by the aforementioned examples, axial flow fans can be implemented on a variety of scales from small to large and for a number of roles. The Polkowice-Sieroszowice Mine located in Poland exhibits an impressive example of one of the largest main axial flow fan installations in the world. This mine operates at a depth of 4,000 ft below the surface. The history of the mine dates back to 1962, and it is now the largest deep-mining project in Europe, producing a total of 12 million tpy.

In 2016, a new ventilation shaft project was completed, that includes an impressive and complex fan installation. Four parallel 4,000 kW fans extract a total of 2,500,000 cfm from the mine at a pressure greater than 28 inches of water gauge. The fans are capable of reversing on- the-fly, and providing up to 60% of the normal operating flow in the reverse direction. Due to the presence of Radon gas in the exhaust airstream, the diffuser towers extend vertically over 100 ft and house filtration units for air scrubbing.

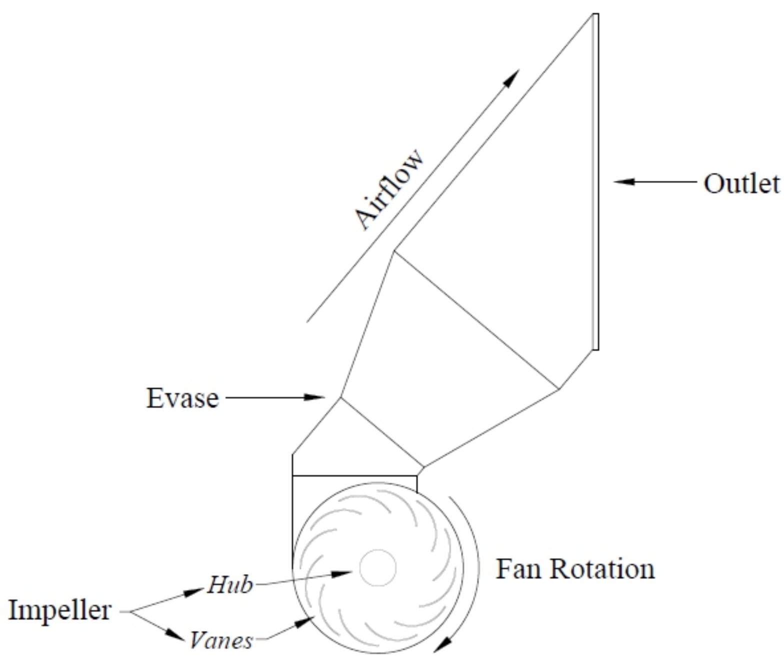
The entire installation also operates within extreme noise constraints, limiting the ambient sound-pressure levels coming from site to less than 35 dB (approximately equivalent to a human whisper at a distance of greater than 10 feet). This was achieved through the use of concrete ducting covered with resonator stones throughout the installation. With a total of 16MW of installed fan power within a relatively compact footprint, operating within a noise limit of 35 dB represents the limits of what is possible for surface fan installations; but in some cases, this may make the difference between obtaining an operating permit and not. The figure below presents an elevated visual overview of this axial flow installation.

Figure 2.4. Elevated visual view of an axial fan installation.

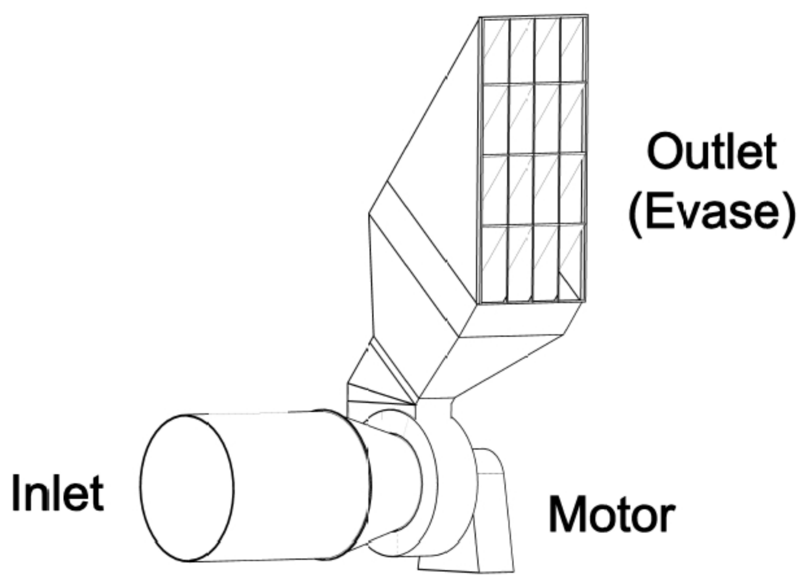
# 2.2 Centrifugal

Centrifugal fans are one of two major fan types utilized in underground mines. This type of fan is composed of a bladed centrifugal impeller in which the inlet is located at the center of the impeller perpendicular to rotation and exits radially from the outer edges. The blades of the impeller are available in a variety of designs, such as curved, straight, reversed curved, etc., to account for the operating conditions experienced in mines. The surrounding fan casing not only guides the inlet flow to the center of the impeller but also guides the exhaust flow from the impeller to a single outlet. This impeller design coupled with the overall layout of the centrifugal fan allows the fan to withstand the cavitation caused by stall conditions for longer periods of time relative to axial flow fans thereby adding a certain amount of robustness to centrifugal fans.

The flow exiting a centrifugal impeller is chiefly radial, a characteristic that easily distinguishes the centrifugal fan from the flow generated by axial flow fans. As the air moves outwards, it is subjected to the centrifugal forces imparted by the rotating impeller. As such, the air being discharged immediately form the fan contains both a rotational velocity component in the direction of the fan's rotation and a radial velocity component. Centrifugal fans are chiefly applied in the U.S. as bleeder exhaust fans because of their ability to generate high fan pressures to overcome these generally high resistance flow zones. However, centrifugal fans can also be applied as main mine fans as long as flow characteristics are within the rated operation of a given fan. From a sound perspective, centrifugal fans in this role can operate much quieter than the typical axial flow fan. For example, if a suitable series of centrifugal fans had been evaluated and were available at the time of the Polkowice-Sieroszowice Mine fan project presented in the Axial Flow section of this topic, the mine may have realized significant cost savings from no longer needing the level of concrete sound mitigation infrastructure that was eventually implemented. This characteristic has prompted many major operations to adopt centrifugal fans as main fans. A conceptual schematic is provided below. In this figure, air enters the fan into the page through the inlet and exits out toward the right from the outlet.



The following figure a conceptual layout of how a centrifugal fan may be implemented as a main mine fan from the inlet diffuser to the outlet evase. The fan motor and motor shaft can be seen attached to the impeller to the right of the figure. This conceptual layout mimics the design and rotation displayed in the schematic figures shown above.



# 3.0 Introduction

The following model provides an overview of how pressures in a fan are defined for quantitative analysis as well as how these values are collected and applied. Compressibility effects are taken into consideration for the computation of fan efficiency in this module because of its noticeable impact on operating costs. Although compressibility effects are specifically presented in this module, the approach for calculating fan efficiency on incompressible flow is also presented as an alternative.

# 3.1 Fan Pressure

The concepts of total, static and velocity pressures for air flow are still applicable fans but in a slightly modified context. The following definitions represent how these three pressure types are treated in terms of mine ventilation fans.

1. Fan total pressure: the increase in total pressure across the fan. Using a pitot static tube facing against the direction of airflow, the total pressure on both the inlet and exhaust sides of the impeller are measured and then the difference in the total pressures is used to determine the increase in the total pressure across the fan, which is the fan total pressure.
2. Fan velocity pressure: the average velocity pressure at the fan outlet. A pitot static tube is used to measure the velocity pressure across multiple points covering the entire surface area of the fan outlet. These individual velocity pressure are then averaged to determine the fan velocity pressure.
3. Fan static pressure: the difference between the fan total pressure and the fan velocity pressure.

Fan pressures are defined in this manner to specifically identify the useful mechanical energy that is imparted by a given fan, which can be deduced from the fan static pressure. The fan static pressure can be used in this manner because the fan velocity pressure is considered a complete loss of useful energy, which does occur if the fan discharges directly to atmosphere. The fan total pressure then represents the full increase in mechanical energy produced by the fan. Based on these definitions, the difference between the total amount of mechanical energy imparted by the fan and the loss of mechanical energy across the fan equals the useful mechanical energy applied to the system by the fan.

Fan manufacturers publish fan characteristic curves in terms of fan static pressure instead of fan total pressure. Since fan static pressure is the difference between fan total pressure and fan velocity pressure, all pressure losses created by inlet and outlet shapes and lengths have already been included. As a result, manufacturers do not need to publish the types of inlet and outlet duct fittings or the conditions at the inlet and outlet, which they would not be able to controls anyways during installation. However, fan velocity pressures are published at times. In these cases, the manufacturers will provide a description regarding the outlet as well as indicate the specific location at the outlet from which the data was collected. This data point is usually gathered either at the fan hub or at the outlet of the evasee.

# 3.2 Fan Survey

The most effective way to determine the performance of a fan is through regularly scheduled fan surveys where data, such as impeller speed, are collected and analyzed. Various analytical techniques can then be applied to the gathered data to quantify fan performance characteristics. These characteristics can be used to ensure that the fan is operating as expected and within design tolerances as well as to create a performance profile for the specific applications of the fan (i.e., fan curves that represent the characteristics of the site rather than the manufacturer's laboratory). These performance benchmarks can be used in conjunction with the fan laws to predict performance at different operating points when needed. Additionally, the collection of historic data through regularly scheduled surveys will also provide the ability to identify any errant data points, which may indicate an impending mechanical failure or an unexpected artifact in the ventilation system, such as a disabled regulator or a short-circuit caused by an unexpected open ventilation control. Although useful, field fan surveys are usually restricted to a narrow sampling of a fan's in-place operating range. Field surveys can also be difficult to execute depending on the design and location of the fan emplacement. The following sections provide an overview of a fan survey including a description of compressibility and efficiency as well as how to gather performance data from an active fan in the field.

# 3.2.1 Compressibility

Any gas is compressible and subject to compressibility effects in a dynamic environment. However, the flow through a fan system can be considered incompressible in the majority of situations, which greatly simplifies the associated computations. Although fan air flow is still subject to compressibility, the error produced by ignoring these effects in pressure calculations is less than 1% for flow with Mach numbers below 0.2. Under certain circumstances with the application of mean air densities, acceptable calculations can even be made up to Mach numbers of 1.0. Given the limited applicability of compressibility to most fan installations, only a brief overview of compressibility will be provided in this section. If a desired application is expected to be subject to significant, complex compressibility effects, academic material specifically covering this subject matter should be consulted.

As previously discussed in this module, airpower was defined as the product of fan static pressure and quantity with the assumption of incompressible flow. This assumption will translate into an error of <1% up to fan pressure of 2.8 kPa or to a Mach number of 0.2 in pressure and performance computations. Although insignificant in many cases, fan operating costs at large mines may be significantly affected by even a 1% error. Additionally, mine fan pressures exceeding 6 kPa can occur especially with fan designed to serve large or high resistance mines. In these circumstances, compressibility should be considered when determining fan performance.

The steady flow equation derived from Bernoulli's equation and modified for the how pressures are treated in terms of fans is provided below where "W" is the impeller shaft work in J/kg of air, "V" is the specific volume of air in m3/kg, "P" is the absolute barometric pressure in Pa, "F" is the frictional losses between the inlet and the outlet in J/kg, "H" is enthalpy in J/kg, and "q" is the heat added to the system through the fan casing between the inlet and the outlet in J/kg. The subscripts "in" and "out" refer to the fan inlet and outlet respectively.

$$\frac{u_{in}^2-u_{out}^2}{2}+(Z_{in}-Z_{out})g+W=\int_{in}^{out}V\mathrm{d}P+F_{in\_out}=(H_{out}-H_{in})-q_{in\_out}$$

The change in elevation across the fan is assumed to be zero, which will be the case in the majority of situations. The change in air velocity across the fan impeller is assumed to be negligible relative to other variables. The velocity assumption makes the increase in total pressure across the fan, or the fan total pressure, equal to the increase in barometric pressure across the fan. Using these assumptions, the velocity and elevation terms, can be eliminated from the fan steady flow equation. The resulting relationship is displayed below.

$$W=\int_{in}^{out}V\mathrm{d}P+F_{in\_out}=(H_{out}-H_{in})-q_{in\_out}$$

The heat transferred through the fan casing from the surrounding environment is not significant in most cases compared to the energy imparted by the fan shaft, which then allows the removal of the heat energy term. The resulting equation is displayed below.

$$W=\int_{in}^{out}V\mathrm{d}P+F_{in\_out}=(H_{out}-H_{in})$$

Thermodynamic efficiency for the fan impeller can only be defined if a point of reference from a fan with no losses can be established. Using such a reference, the difference between the fan understudy and a theoretically perfect fan can be used to determine efficiency. A perfect fan would not be subject to any frictional losses and thus would not have any heat losses into or out of the system, which allows the friction term to be removed. The heat enthalpy changes to compare the difference between the perfect, isentropic system to the heat enthalpy of the target fan. The resulting equation is displayed below.

$$W=\int_{in}^{out}V\mathrm{d}P=(H_{isen}-H_{fan})$$

# 3.2.2 Efficiency

Power is delivered to the drive shaft of a fan from a motor and through a transmission. The two major mechanical energy transfer points in the motor and the transmission are the primary sources for energy loss in a fan drive system. In a properly maintained fan electrical and mechanical system, the large majority, approximately 95%, of the consumed electrical energy is converted to mechanical energy in the impeller drive shaft. The impeller then converts most of the mechanical energy into air power, or the rate of work done to produce the movement of air through an increase in pressure induced by the impeller. The energy losses are effectively heat being consumed by impeller friction and air friction across the fan casing, which increases the temperature of the air. Impeller efficiency, $\eta$, may be defined as the ratio of air power to shaft power.

$$\frac{Air\ Power}{Shaft\ Power}$$

The overall efficiency of the fan mechanical system (i.e., the motor, transmission, and impeller) may be defined as the ratio of air power to motor input power.

$$\frac{Air\ Power}{Motor\ Input\ Power}$$

The following will provide an overview of fan efficiency with specific concentration on demonstrating the relationship between power and the pressure-volume duty achieved by the fan. This particular relationship directly relates to the operating cost of the fan. Two main methods are employed when determining the efficiency of a fan. These methods are the pressure-volume method and the thermometric method. The pressure-volume methods is, by a significant margin. applied the most frequently and is the most widely accepted approach. This method utilizes measurements of fan pressure, airflow, and shaft power to determine efficiency.

As previously discussed, assuming incompressible flow is convenient for many calculations and is acceptable in many circumstances at the resulting propagated error is about 1%. However, in terms of power, 1% can translate to significant monetary costs. Considering the impact of compressibility on air power and thus on efficiency, the following discussion will take compressibility into account. However, a derivation of the power equations that include compressibility will not be discussed.
If compressibility is not of significant interest, then the aforementioned relationships between air power and shaft power as well as air power and motor input power can be used to determine fan efficiency directly.

The following equation can be used to determine the isentropic air power, or rate of energy consumed by the impeller from a steady-flow system (i.e., system with equal entropy), of a fan as a function of fan pressure, $p_f$, air quantity at the inlet, $Q_{in}$, and the compressibility coefficient, K.

$$P_{isen} = p_f \cdot Q_{in} \cdot K$$

K can be calculated using the equation below where $\gamma$ is the isentropic index, $p_B$,in is the absolute barometric pressure at the inlet, $p_{B,out}$ is the absolute barometric pressure at the outlet, and $p_f$ is the fan pressure, which is usually fan static pressure but can change depending on the type of data that was gathered or the purpose of gathering that data. The isentropic index, $\gamma$, for air is the ratio of the specific heat capacity of air in a fixed pressure system to the specific heat capacity of air in a fixed volume system, or $\gamma = c_p/c_v$. The isentropic index can thus be either calculated or referenced from a table or graph containing values representing a variety of conditions. For unsaturated, dry air, which is also called standard air, $\gamma$ is equal to 1.4.

$$K = \frac{\gamma}{\gamma - 1} \cdot \frac{p_{B,in}}{p_f} \left[ \left( \frac{p_{B,in}}{p_{B,out}} \right)^{1-\frac{1}{\gamma}} - 1 \right]$$

K can also be defined as a ratio of inlet and outlet absolute barometric pressures. The resulting equation is displayed below. This equation can be simpler to apply in some cases when a K vs. Absolute Pressure Ratio graph can be referenced for the desired calculation.

$$K = \frac{\gamma}{\gamma - 1}\left[\frac{\left(\frac{p_{B,out}}{p_{B,in}}\right)^{\frac{\gamma-1}{\gamma}} - 1}{\frac{p_{B,out}}{p_{B,in}} - 1}\right]$$

Based on the previously introduced relationship between fan impeller efficiency and air power/shaft power, the following equation can be derived to include isentropic air power.

$$\eta_{isen} = \frac{P_{isen}}{Shaft\ Power} = \frac{p_f \cdot Q_{in} \cdot K}{Shaft\ Power}$$

The terms "static efficiency" and "total efficiency" may be encountered. These names simply refer to the utilization of fan static pressure or fan total pressure, respectively, to calculate impeller efficiency.

# Gaseous Contaminants

[Jump to Today](#) ✎ Edit

## Gaseous Contaminants

Gaseous contaminants are one of the most hazardous substances encountered in mine ventilation.  They may be natural (e.g., strata gases) or manmade (e.g., chemical operations), and they can present a host of risks, including explosivity and toxicity to humans.  This module will address the occurrence, effects, and control of typical mine gases including oxygen, carbon dioxide, hydrogen sulfide, sulfur dioxide, carbon monoxide, and methane, to name just a few.  Their detection is also addressed here.  While individual gases that may occur with diesel particulate matter (DPM) are addressed here, their net effects and control are more thoroughly addressed in the DPM module.  Finally, radiation associated with underground mining is addressed here.  Specific handling of radioactive material is not addressed, but effects and engineering controls specific to ventilation are included.

**Learning Objectives**

1. **Describe the gases that occur in mines, including typical sources and effects.**
2. **Explain the technology used to detect various gases, including sampling and analysis techniques.**
3. **Articulate the engineering controls used for the control of various gases.**
4. **Utilize case studies to describe the importance of rigorous ventilation control for gases, as well as to articulate the risks they pose.**
5. **List types of radiation emissions and the efficacy of basic controls.**
6. **Express the health effects of radiation as well as monitoring methods.**

Get Started

Go to the Course Index

About the Aeolus Project

## Course Summary:

| Date | Details |
| --- | --- |

# 1.0 Introduction to Mine Gases

Atmospheric gases are routinely found in mines as part of the ventilation air.  A typical atmosphere includes:

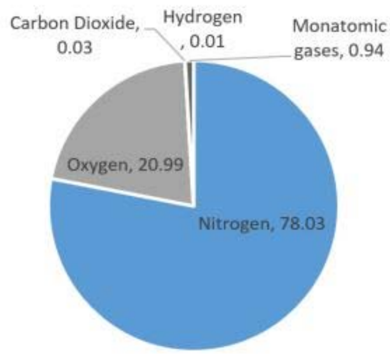



**Figure 1.1. Makeup of standard air**

It is important to note that composition by volume can vary around the world.  Mines with outcropping or portals on farms with livestock may see increased levels of methane.  Also carbon dioxide levels are changing and can vary by location (**NOAA, 2017 (https://canvas.instructure.com/courses/1094345/pages/resources)** ).

Additionally, many mines contain gas-bearing strata.  Coal mine strata typically release methane and ethane, salt and trona mines may contain also contain methane, and metal/nonmetal mines can contain sulfur hydroxide and hydrogen sulfide.  Release rates and quantities are usually a function of strata permeability, reservoir pressure, mining rate, and barometric pressure.  Release of strata gases may be fairly continuous, with mostly constant rates or much more irregular with large releases, and low frequency.  An understanding of strata gas occurrence and detection is critical for every ventilation engineer.

Finally, the dangers these gases pose include explosion, toxicity, and other health risk (e.g., radioactivity - carcinogen).

The following gases are routinely found in mines, and summarized below. Click on each gas to learn more:

**Table 1.1. Summary of Common Mine Gas Properties**

| Name | Symbol | Density relative to dry air* | Primary source in mines | Hazard | Smell, color, taste | MSHA TLV | Detection** | Flammability limits in air |
|---|---|---|---|---|---|---|---|---|
| Dry air | - | 1.000 | ventilation air | none | none | - | - | - |
| **Carbon dioxide (https://canvas.instructure.com/courses/1094345/pages/carbon-dioxide)** | $CO_2$ | 1.529 | ventilation air, fires, explosions, oxidation of carbon, internal combustion engines, blasting, respiration | Increased respiration | slight acid taste and smell | TWA=0.5%<br>STEL=3.0% | optical infrared | - |
| **Carbon monoxide (https://canvas.instructure.com/courses/1094345/pages/carbon-monoxide)** | CO | 0.967 | fires, explosions, oxidation of carbon, internal combustion engines, blasting, respiration, spontaneous combustion | Toxic, Explosive | none | TWA=50 ppm<br>STEL=400 ppm | electrochemical, catalytic oxidation, semi conductor, infrared | 12.5-74.2% |
| **Ethane (https://canvas.instructure.com/courses/1094345/pages/ethane)** | $C_2H_6$ | 1.049 | strata | Explosive | none | - | - | 3-12.4% |
| **Hydrogen (https://canvas.instructure.com/courses/1094345/pages/hydrogen)** | $H_2$ | 0.0696 | combustion, battery charging (less common with new battery technology) | Explosive | none | - | catalytic oxidation | 4-74% |
| **Hydrogen sulfide (https://canvas.instructure.com/courses/1094345/pages/hydrogen-sulfide)** | $H_2S$ | 1.190 | strata, decomposition of organic materials, acid water on | Toxic, Explosive | rotten egg smell | TWA=10 ppm<br>Ceiling=15 ppm | electrochemical semiconductor | 4.3-45.5% |

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | | | sulfides, stagnant water | | | | | |
| **Methane** (https://canvas.instructure.com/courses/1094345/pages/methane) | CH4 | 0.554 | strata | Explosive | none | 1% (isolate electricity)<br>2% (remove personnel) | catalytic oxidation, thermal conductivity, optical, acoustic | 5-15% |
| **Nitrogen** (https://canvas.instructure.com/courses/1094345/pages/nitrogen) | $N_2$ | 0.967 | ventilation air, strata | Inert | none | - | by difference | - |
| **Nitric oxide** (https://canvas.instructure.com/courses/1094345/pages/oxides-of-nitrogen) | NO | 1.037 | internal combustion engines, blasting, welding | Toxic (oxidizes to $NO_2$) | irritating to eyes, nose, and throat | TWA=25 ppm | electrochemcial, infrared | - |
| **Nitrous oxide** (https://canvas.instructure.com/courses/1094345/pages/oxides-of-nitrogen) | $N_2O$ | 1.519 | internal combustion engines, blasting, welding | Incapacitating (laughing gas) | sweet smell | TWA=50 ppm | electrochemical | - |
| **Nitrogen dioxide** (https://canvas.instructure.com/courses/1094345/pages/oxides-of-nitrogen) | $NO_2$ | 1.588 | internal combustion engines, blasting, welding | Toxic | reddish brown, acidic smell and taste | TWA=3 ppm Ceiling =5 ppm | electrochemical infrared | - |
| **Oxygen** (https://canvas.instructure.com/courses/1094345/pages/oxygen) | $O_2$ | 1.105 | air | Explosive (with reactive gases); Oxygen deficiency | none | >19.5% | electrochemical, paramagnetic | - |
| **Radon** (https://canvas.instructure.com/courses/1094345/pages/what-is-a-working-level-wl) | Rn | 8.075 | uranium minerals in strata | Radioactive | none | 1 **WL** (https://canvas.instructure.com/courses/1094345/pages/what-is-a-working-level-wl) and 4 **WL-months** (https://canvas.instructure.com/courses/1094345/pages/what-is-a-working-level-wl) per year | radiation detectors | - |
| **Sulfur Dioxide** (https://canvas.instructure.com/courses/1094345/pages/sulfur-dioxide) | $SO_2$ | 1.891 | oxidation of sulfides, acid water on sulfide ores, internal combustion engines | Toxic | acid taste, suffocating smell | TWA=2 ppm<br>STEL=5 ppm | electrochemical, infrared | - |

*at standard temperature and pressure (32 degrees F {0 degrees C} and 30 in. Hg {101.325 kPa})

**Most of these gases may be detected by **gas chromotography (or mass spectrometry) or stain tube** (https://canvas.instructure.com/courses/1094345/pages/methods-of-gas-detection) .

(Modified from **McPherson, 2009** (https://canvas.instructure.com/courses/1094345/pages/resources) . (https://canvas.instructure.com/courses/1094345/pages/resources) )

# Carbon Dioxide

**Molecular Formula:** $CO_2$

**Molar Mass:** 44.009 g/mol

**Hazard:** Asphyxiant

***Exposure Limits***

| Standard* | TWA | STEL |
|---|---|---|
| ACGIH | 0.5% | 3.0% |
| NIOSH | 0.5% | 3.0% |
| OSHA | 0.5% | - |
| MSHA (coal) | 0.5% | 3.0% |
| MSHA (M/NM) | 0.5% | 1.5% |

**About Threshold Limit Values (TLVs) (https://canvas.instructure.com/courses/1094345/pages/about-threshold-limit-values-tlv)**

Exposure symptoms include: headache, dizziness, restlessness, paresthesia; dyspnea (breathing difficulty); sweating, malaise (vague feeling of discomfort); increased heart rate, cardiac output, blood pressure; coma; asphyxia; convulsions

Carbon dioxide is heavier than air and tends to pool near the floor.

Carbon dioxide is a product of combustion and human respiration.  Carbon dioxide may occur as a strata gas, in which case emissions and even outbursts may pose health and safety hazards to miners.  Outbursts of carbon dioxide in coal mines have been documented in Australia, Czechoslovakia, France, Poland, Turkey, and the UK (**Beamish and Crosdale, 1998 (https://canvas.instructure.com/courses/1094345/pages/resources)** ).

**US standards are provided and include: the American Conference of Governmental Industrial Hygienists (ACGIH), the National Institute for Occupational Safety and Health (NIOSH), the Occupational Safety and Health Administration, and the Mine Safety and Health Administration (MSHA).*

# Carbon Monoxide

**Molecular Formula:** CO

**Molecular Mass:** 28.01 g/mol

**Hazard:** poisonous and explosive

***Exposure Limits***

| Standard* | TWA | STEL | C |
|---|---|---|---|
| ACGIH | 25 ppm | - | - |
| NIOSH | 35 ppm | - | 200 ppm |
| OSHA | 50 ppm | - | - |
| MSHA (coal) | 50 ppm | 400 ppm | - |
| MSHA (M/NM) | 50 ppm | 400 ppm | - |

(**NIOSH, 2016) (https://canvas.instructure.com/courses/1094345/pages/resources)**

**About Threshold Limit Values (TLVs) (https://canvas.instructure.com/courses/1094345/pages/about-threshold-limit-values-tlv)**

Carbon monoxide is poisonous, colorless, odorless and tasteless. In the human body carbon monoxide binds to hemoglobin, producing carboxyhemoglobin, inhibiting the transport of oxygen. Hemoglobin has an affinity for carbon monoxide that is more than 200 times greater than its affinity for oxygen.

Exposure symptoms include: headache, dizziness, decreased pulse and respiratory rates, unconsciousness, and death.

Carbon monoxide is a product of incomplete combustion, and is often the cause of death or serious injury in mine fires. Tolerance for carbon monoxide (like most gases with health effects) is varied, but exposure and comparison of exposure time, respiration rates and blood saturation is given by McPherson, which will give the reader a general idea of effects over time.

(**McPherson, 2009 (https://canvas.instructure.com/courses/1094345/pages/resources)** ).

**US standards are provided and include: the American Conference of Governmental Industrial Hygienists (ACGIH), the National Institute for Occupational Safety and Health (NIOSH), the Occupational Safety and Health Administration, and the Mine Safety and Health Administration (MSHA).*

# Ethane

**Molecular Formula:** $C_2H_6$

**Molar Mass:** 30.07 g/mol

**Hazard:** Explosive (3 to 12.4% in air)

Ethane is not usually encountered in the explosive range in underground coal mines, although it does sometimes coexist as a seam gas with methane (**Finfinger and Cervik, 1979 (https://canvas.instructure.com/courses/1094345/pages/resources)** ) and may be emitted in the same manner.  It may also be emitted from other strata or as a product of a mine fire. In cases where ethane is present care must be taken to ensure that it is maintained well below the lower explosive limit (LEL).

# Hydrogen

**Molecular Formula:** $H_2$

**Molar Mass:** 1.00794 g/mol

**Hazard:** Explosive (4% to 74.2% in the presence of low oxygen to normal air)

Hydrogen is produced by the incomplete combustion of carbon materials during fires and explosions. It may also be liberated when water or steam comes in contact with hot carbon materials during firefighting. Battery charging also produces hydrogen. (**MSHA, 2008 (https://canvas.instructure.com/courses/1094345/pages/resources)** )

May be found around battery charging stations, after blasting, after an active fire or explosion.

# Hydrogen Sulfide

**Molecular Formula:** $H_2S$

**Molar Mass:** 34.08088 g/mol

**Hazards:** Toxic and Flammable

***Exposure Limits***

| Standard* | TWA | STEL | C |
|---|---|---|---|
| ACGIH | 1 ppm | 5 ppm | - |
| NIOSH | 10 ppm | - | - |
| OSHA | 10 ppm | 20 ppm | 50 ppm |
| MSHA (coal) | 10 ppm | 20 ppm (5 min) | see STEL |
| MSHA (M/NM) | 10 ppm | 20 ppm (5 min) | see STEL |

[About Threshold Limit Values (TLVs) (https://canvas.instructure.com/courses/1094345/pages/about-threshold-limit-values-tlv)](https://canvas.instructure.com/courses/1094345/pages/about-threshold-limit-values-tlv)

Hydrogen sulfide is considered immediately dangerous to life and health (IDLH) at 100 ppm by the AIGCH.

Exposure to hydrogen sulfide may cause the following symptoms: irritation eyes, respiratory system; apnea, coma, convulsions; conjunctivitis, eye pain, lacrimation (discharge of tears), photophobia (abnormal visual intolerance to light), corneal vesiculation; dizziness, headache, lassitude (weakness, exhaustion), irritability, insomnia; gastrointestinal disturbance. Exposure may be through contact with the skin or eyes, or via inhalation (NIOSH pocket guide). Symptoms may be delayed ([PubChem, 2016 (https://canvas.instructure.com/courses/1094345/pages/resources)](https://canvas.instructure.com/courses/1094345/pages/resources) ).

Hydrogen sulfide is a colorless gas that is detectable by the human nose beginning at about 0.2 ppm. It has a distinctive rotten egg smell, but, at levels above 100 ppm, olfactory fatigue occurs, and the smell is no longer detectable. Hydrogen sulfide is sometimes referred to as "stinkdamp".

**US standards are provided and include: the American Conference of Governmental Industrial Hygienists (ACGIH), the National Institute for Occupational Safety and Health (NIOSH), the Occupational Safety and Health Administration, and the Mine Safety and Health Administration (MSHA).*

**Case Study: Workers overcome by hydrogen sulfide**

"Seven men died Monday, April 12, 1971, as a result of exposure to hydrogen sulfide gas in advance workings on the 800-foot level of the Barnett Complex Mine, Ozark-Mahoning Company, Pope County, Illinois (U/G Fluorspar Lead Zinc). Drifting and test drilling operations to locate an ore vein were being conducted at the extreme end of the 800-foot south level, on Friday, April 9. Near the end of the day shift, the third of three test holes struck a watercourse and water under high pressure was released into the drift. Work was discontinued in the area, and the water allowed to flow into the drift on the belief the body of water would soon be drained. Reportedly, hydrogen sulfide was not liberated on Friday; the two workers who were drilling did not smell the gas or suffer eye irritation. The presence of hydrogen sulfide gas was first detected during the day shift on Saturday, when two miners, out of curiosity, went to the face to look at the water flow. The miners reported that the gas irritated their eyes and caused "tightness" in their chests. At some time between the end of the second shift on Saturday and Monday morning, one of three fans in the auxiliary ventilation system for the 800 south level failed. What ventilation existed at the south end of the 800-foot level thereafter is unknown. On Monday, April 12, installation of a replacement fan was completed shortly after noon. Before the fan was started, a miner went inby the fan to obtain measuring sticks. He was seen by the men installing the fan, but testimony is not clear as to his being aware of or warned of a potential danger. In about a half hour, the miner's brother went into the area to look for him. When neither of the two men returned, other miners, without respiratory protection, attempted rescue. At this time the replacement fan was started. In the course of events, five additional miners were overcome while several others, although repeatedly entering the drift and being affected by the gas, did escape by cutting into the ventilation tubing for fresh air." (The Southeast Missourian, 1971.)

# Methane

**Molecular Formula:** $CH_4$

**Molar Mass:** 16.04 g/mol

**Hazard:** Explosive (5 to 15% in air)

***Exposure Limits: per MSHA regulation, de-energize equipment at 1% and take corrective action, withdraw workers at 2%.***

Methane is lighter than air and is generally found near the roof or where emitting from the strata. Emissions may be fairly constant over a coal face or occur in "bleeders" or "pipes" - pathways of increased permeability. Methane can also flow in from the roof or floor. High levels of methane are sometimes referred to as "firedamp".

Methane is primarily considered a hazard in coal mines, but can certainly occur in other deposits including salt, trona, and potash. In coal, methane exists as a free gas and an adsorbed gas. Coal surfaces attract molecules of methane, carbon dioxide, nitrogen, and water vapor. The release of methane is dominated by two types of flow:

Diffusion: takes place in the micropore structure and occurs when a difference in concentration of molecules of a given gas occurs.

Darcy Flow: Laminar flow that is dominant in the cleat structure.

When considering methane control, important parameters to consider are permeability, reservoir pressure, and in situ gas content. It is also important to note that falling barometric pressure can create an increased pressure differential between the underground opening and the coal seam such that methane emissions may increase when barometric pressure falls.

Ventilation methods for methane control include dilution by ventilation air and degasification prior to mining. Degasification has substantially improved safety in gassy underground coal mines, and a number of methods have been developed which include drilling from the surface and in-mine drilling. Several methods are briefly described below.

Figure 1. A drill rig for a vertical-to-horizontal multilateral well at an underground coal mine in China.

**Horizontal wells (in mine)**

In seam horizontal drilling can include short hole (<500 feet) and long hole (thousands of feet). A specialized underground drilling rig is utilized. Advantages and disadvantages of horizontal wells are given below:

| Advantages | Disadvantages |
|---|---|
| Avoids surface disturbance | <u>Logistics</u><br><br>Cuttings disposal<br>Ventilation<br>Moving methane via in mine pipeline |
| Short hole can be accomplished "in house" with trained miners | Infrastructure and associated risk |
| No drilled footage to access the seam | Relatively short degasification time (for short hole) |

**Traditional vertical wells**

Traditional vertical wells are relatively inexpensive to drill but require more roads and a more extensive gathering system which becomes expensive in very rugged terrain. They also leave a larger net footprint due to more drill sites. However, it is possible to convert exploration wells into production wells.

**Vertical-to-horizontal wells and multilateral wells**

Vertical-to-horizontal wells are the most challenging to drill, more so in thin seams. The advantage is that a large area may be access with relatively little surface disturbance, and the degasification prior to mining can be quite long. There have occasionally been issues in ensuring that the wells are adequately plugged prior to mining because the horizontal section may undulate, but this is not a significant issue. Common patterns are shown below. Orientation with regard to cleat structure and stress may impact production, and like other wells, these may be hydraulically fractured and treated with proppant to enhance permeability.

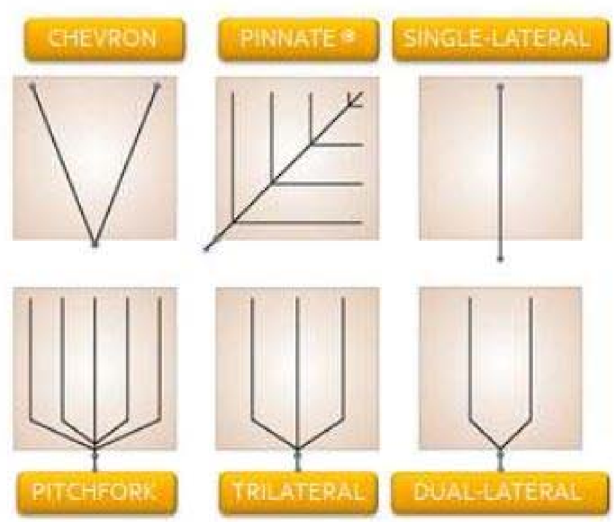



Figure 2. Plan view of common vertical-to-horizontal multilateral well patterns. Variations of the trilateral are also referred to as a "turkey foot".

**Vertical gob wells**

Also known as gob gas ventholes (GGV), these are simple vertical wells, often with a larger diameter. The well is usually drilled to within about 200 feet of the seam. It is cased to above the caving zone and then a slotted pipe is installed. GGVs take advantage of caving to capture gas and allow for better control of gas in longwall systems. Methane concentrations can vary considerably and may not be pipeline quality at the wellhead.

**Outburst hazards**

Methane outbursts (not to be confused with *rockbursts*) refer to violent inundation of methane into a mine working. The one common feature is the existence of mechanically weakened pockets of mineral within the seam containing gas at high pressure. Outbursts pose a danger due to flying material, asphyxiation, and ignition. They can be very difficult to predict.

# Nitrogen

**Molecular Formula:** $N_2$

**Molar Mass:** 14.0067 g/mol

**Hazard:** Inert

Nitrogen is an inert gas that exists in standard air at 78% by volume.  It is sometimes used to inert sealed areas to prevent spontaneous combustion or extinguish fires.

# Oxides of Nitrogen

**Molecular Formula:** $NO_2$, NO

**Molar mass:** 46.0055 g/mol, 30.01 g/mol

**Hazards:** Toxic

***Exposure Limits for $NO_2$***

| **Standard*** | **TWA** | **STEL** | **C** |
|---|---|---|---|
| ACGIH | 3 ppm | 5 ppm | - |
| NIOSH | - | 1 ppm | - |
| OSHA | | 1 ppm | - |
| MSHA (coal) | - | - | 5 ppm |
| MSHA (M/NM) | - | - | 5 ppm |

***Exposure Limits for $NO_2$***

| **Standard*** | **TWA** | **STEL** | **C** |
|---|---|---|---|
| ACGIH | 25 ppm | - | - |
| NIOSH | 25 ppm | - | - |
| OSHA | 25 ppm | - | - |
| MSHA (coal) | 25 ppm | - | - |
| MSHA (M/NM) | 25 ppm | 37.5 ppm | - |

Oxides of Nitrogen include Nitrogen dioxide, nitrous oxide and nitric oxide. **About Threshold Limit Values (TLVs) (https://canvas.instructure.com/courses/1094345/pages/about-threshold-limit-values-tlv)**

When mixed with water, oxides of nitrogen form acids.  Because of this reaction they are particularly damaging to the lungs and the effects may not be apparent for several hours after exposure. Often, such symptoms don't show up until several hours after you're exposed to the gas. Exposure to .01 to .015 percent (100 to 150 ppm) can be dangerous for even short exposures, and .02 to .07 percent (200 to 700 ppm) can be fatal for short exposures (MSHA, 2008).

Nitric oxide (NO) does not exist in large amounts in the air because it quickly combines with oxygen (oxidizes) to form nitrogen dioxide (NO2).

Oxides of nitrogen are colorless at low concentrations and become reddish-brown at higher concentrations. They smell and taste like blasting powder fumes.

Oxides of nitrogen are produced by burning and by the detonation and burning of explosives. They are also emitted from the exhaust of diesel engines. In the presence of electrical arcs or sparks, nitrogen in the air combines with oxygen (oxidizes) to form oxides of nitrogen (MSHA, 2008).

**US standards are provided and include: the American Conference of Governmental Industrial Hygienists (ACGIH), the National Institute for Occupational Safety and Health (NIOSH), the Occupational Safety and Health Administration, and the Mine Safety and Health Administration (MSHA).*

*(**Cauda, 2012 (https://canvas.instructure.com/courses/1094345/pages/resources)** ; **NIOSH, 2016 (https://canvas.instructure.com/courses/1094345/pages/resources)** )*

# Oxygen

**Molecular Formula:** $O_2$

**Molar Mass:** 31.999 g/mol

**Hazard:** Deficiency can result in negative health effects and death, it also increases combustibility.

***Exposure Limits***

| Standard* | minimum |
|---|---|
| ACGIH | 18% |
| NIOSH | 19.5% |
| OSHA | 19.5% |
| MSHA (coal) | 50 ppm |
| MSHA (M/NM) | 50 ppm |

Oxygen is essential for human life, and generally constitutes 20.9% of the atmosphere by volume. It is odorless and colorless, and reacts with combustible and reducing materials, generating fire and explosion hazards. Oxygen deficiency is sometimes referred to as "black damp".

Some exposure symptoms may be present at very high concentrations, generally not encountered in mines. Low levels of oxygen will produce the following human health effects:

**Human effects of oxygen depletion** (McPherson, 2009.)

| % Oxygen in Air | Effect |
|---|---|
| 17 | Faster, deeper breathing |
| 15 | Dizziness, buzzing in ears, rapid heartbeat |
| 13 | Possible loss of consciousness |
| 9 | Fainting, unconsciousness |
| 7 | Life endangerment |
| 6 | Convulsive movements, death |

**US standards are provided and include: the American Conference of Governmental Industrial Hygienists (ACGIH), the National Institute for Occupational Safety and Health (NIOSH), the Occupational Safety and Health Administration, and the Mine Safety and Health Administration (MSHA). ACGIH discuss physiological limits based on the partial pressure of oxygen in pulmonary capillaries. This can be altitude adjusted, unlike the % concentration.*

# Sulfur Dioxide

**Molecular Formula:** $SO_2$

**Molar Mass:** 63.962 g/mol

**Hazards:** Toxic

***Exposure Limits***

| Standard* | TWA | STEL | C |
|---|---|---|---|
| OSHA | 5 ppm | - | - |
| NIOSH | 2 ppm | 5 ppm | - |

IDLV = 100 ppm (Immediately dangerous to life or health)

**About Threshold Limit Values (TLVs) (https://canvas.instructure.com/courses/1094345/pages/about-threshold-limit-values-tlv)**

**Summary**

Sulfur dioxide is a colorless gas with a characteristic pungent and irritating odor.  It also has an acid taste.  It dissolves easily in water (**PubChem, 2016 (https://canvas.instructure.com/courses/1094345/pages/resources)** )

Sulfur dioxide can occur in mines due to oxidation of sulfides, acid water on sulfide ores, and internal combustion engines (**McPherson, 2009 (https://canvas.instructure.com/courses/1094345/pages/resources)** ). It can also be used for cyanide destruction when the chemical is used for gold extraction.

Sulfur dioxide can be absorbed by inhalation, skin, or eye contact.  Symptoms of exposure include: irritation eyes, nose, throat; rhinorrhea (discharge of thin nasal mucus); choking, cough; reflex bronchoconstriction; cough; shortness of breath. sore throat; labored breathing. (**NIOSH Pocket Guide, 2016 (https://canvas.instructure.com/courses/1094345/pages/resources)** )

(**NIOSH pocket guide, 2016 (https://canvas.instructure.com/courses/1094345/pages/resources)** )

# What is a Working Level (WL)?

Exposure to radiation for miners is calculated working level month (WLM) under US mining regulation ([30 CFR, Part 57, 2017 (https://canvas.instructure.com/courses/1094345/pages/resources)](https://canvas.instructure.com/courses/1094345/pages/resources)):

**§57.5038 Annual exposure limits.**

*No person shall be permitted to receive an exposure in excess of 4 WLM in any calendar year.*

**§57.5039 Maximum permissible concentration.**

*Except as provided by standard §57.5005, persons shall not be exposed to air containing concentrations of radon daughters exceeding 1.0 WL in active workings.*

A WL is 130,000 MeV alpha energy per liter of air

*So what is a WLM*? WLM is exposure to 130,000 MeV alpha energy per liter of air for one month (170 hours [4x40 hour/week +10]), which is approximately 100 pc/l.

It is estimated that the general population is exposed to about 200-300 mrem per year from various sources, including medical, naturally occurring radon, terrestrial, cosmic, fallout, industrial, and internal (e.g., $K^{40}$, $Ra^{226}$, $Pb^{210}$, $Rn^{22}$, $C^{14}$) varying with target organs.

Generally, any industry with radiation hazards should target **ALARA** - doses as low as reasonably achievable ([US NRC, 10CFR, Part 20, 2017 (https://canvas.instructure.com/courses/1094345/pages/resources)](https://canvas.instructure.com/courses/1094345/pages/resources)).

Because the effects of radiation can differ substantially based on epidemiology and exposure, quality factors are often also implemented. For more information on understanding dose and decay see:

[Introduction to Ionizing Radiation (https://www.osha.gov/SLTC/radiationionizing/introtoionizing/ionizinghandout.html)](https://www.osha.gov/SLTC/radiationionizing/introtoionizing/ionizinghandout.html)

# 2.0 Methods of Gas Detection

**Catalytic-oxidation detectors**

These detectors measure combustible gases (e.g., methane and carbon monoxide) via heat generated during oxidation or change in electrical resistivity (Wheatstone Bridge). These detectors are limited. First, they are generally only viable in concentrations from 0% to the lower explosive limit of a gas (LEL), although there are exceptions. They can also be prone to interference from other combustible gases. For example, an methane detector exposed to ethane may give an inaccurate reading. Finally, they are also prone to drift and must be calibrated regularly (generally every 30 days).

**Electrochemical sensors**

Electrochemical sensors may be used to measure oxygen, carbon monoxide, hydrogen sulfide, and oxides of nitrogen. The gas under measurement reacts with an electrode in an electrolyte, and the resulting current is proportional to the gas present.

**Optical detection**

Optical detection can be used for a number of gases and relies on the fact that different gases absorb light at specific and distinct wavelenghs. Measurement of this absorption allows for the calculation of gas concentration. Detectors may be infrared and near infrared. One advantage is they may measure full range or near full range of the gas concentration (0-100%). They typically are not prone to interference from other gases, but may be prone to interference from dust and humidity.

**Electrical conductivity detectors**

Electrical conductivity detectors use semiconductors that change resistance in the presence of the gas under measurement.

**Stain tubes**

Stain tubes generally consist of some kind of reactive media inside a glass tube, and they usually come with a fitted hand pump. The user simply breaks the end of the glass tube, and pulls the air under question through the media with the hand pump. If the gas is present the media will change color and markings on the side of the glass tube will indicate concentration. The glass tube is one time use only, but provides fast and relatively cheap results. It is highly recommended that mines keep stain tubes nearby for emergency gas detection. For instance if your mine has never experience inflow of hydrogen sulfide, but a neighboring mine has or you have reason to suspect that it could happen, then it is sensible to keep at least one hydrogen sulfide detector and some stain tubes on the property.

Figure 2.1. A stain detector tube attached to a hand pump (**OSHA, 2017b (https://canvas.instructure.com/courses/1094345/pages/resources)** ).

**Mass Spectrometry**

In these instruments, the gas sample passes through a field of free electrons emitted from a filament or other source. Collision of the electrons with the gas molecules produces ions, each with a mass/charge ratio specific to that gas. The ions are accelerated by electromagnets and then pass through a magnetic deflection field which separates them into discrete beams according to their mass/charge ratios. The complete mass spectrum can be scanned and displayed on an oscilloscope or the signals transmitted to recorders. (McPherson, .)

## Gas Chromatography

Gas chromatographs are used widely for the laboratory analysis of sampled mixtures of gases.  Portable units are also manufactured. An inert carrier gas is pumped continuously through one or more columns (or coils) which contain gas adsorbents. The latter may be granulated solids or liquids. A small pulse of the sample gas mixture is injected into the line upstream from the columns. The constituent gases are initially adsorbed by the column materials. However, the continued flow of the carrier gas causes subsequent desorption of each gas at a time and rate dependent upon its particular adsorption characteristics. The result is that the gases leave the adsorbent columns as discrete and separated pulses. Their identification and measurement of concentration is carried out further downstream by one or more of the detection techniques described in this section. (McPherson, ).

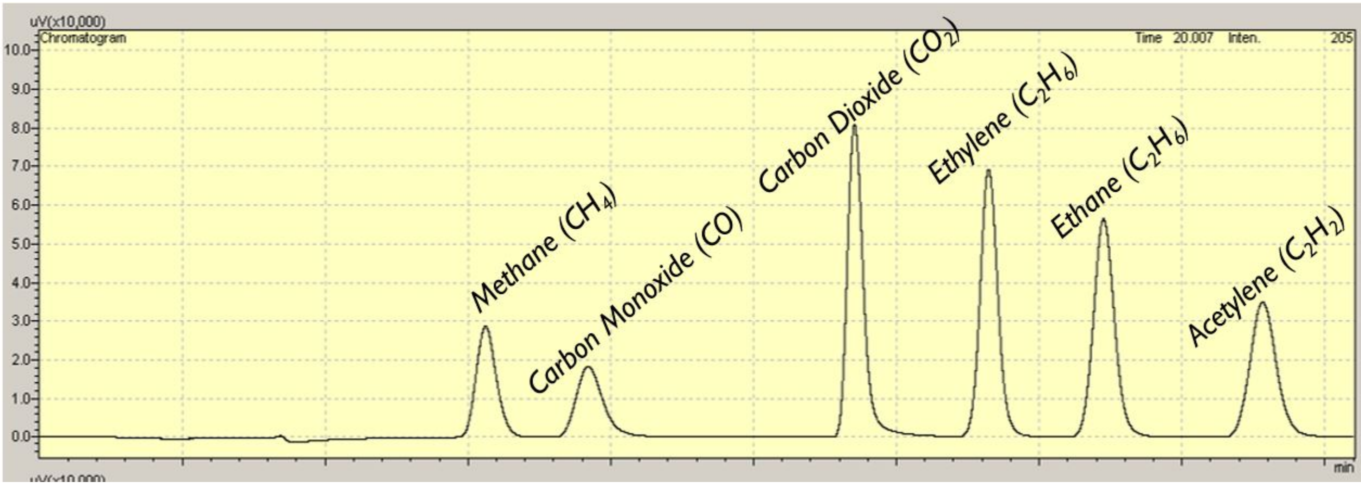



Figure 2.2. Chromatogram of a calibration gas.  Peaks are correlated with known retention times for a particular column and detector configuration.  Areas under the peak are correlated with the known concentrations in the calibration gas.  For an unknown sample, a linear correlation between area of the known peak and the unknown peak versus concentration is assumed - as long as the concentrations are similar.  In other words, several calibration gases may be necessary.

*Mass spectrometry versus gas chromatography - what is the difference?*

A mass spectrometer can tell the user the atomic number of the gas(es) present.  A gas chromatograph simple gives a peak and a retention time which is generally matched to a calibration gas, and the user extrapolates that a particular gas is present if it elutes at the same time as an analyte in the calibration gas.  In other words, with a gas chromatograph the user makes an educated guess as to which gases are present, while a mass spectrometer can tell the user which gases are present.  Mass spectrometry is the more expensive of the two methods, and generally, we have a good idea of what gases may occur in mines.

# 3.0 Levels of Gas Detection

There are multiple levels of gas detection in a mine.  These levels are site specific, and are also often specified by regulation in the US (e.g., handheld and machine mounted methane detectors).  The levels are described below:

1. Personal detection

Personal detection involves the detection of gases by a handheld detector that is generally portable and carried by miners.  These detectors may utilize a number of different technologies.  It is critical in personal detection that the user understands maintenance, care, and limits of the detector.  Has it been calibrated recently?  Does it only measure up to the lower explosive limit?  Additionally, the user must know something about the properties of the gas being measured.  If methane, which is lighter than air, is being measured then the detector should be held near the roof, while heavier gases, like carbon monoxide, may be measured near the floor. (**MSHA, 2008 (https://canvas.instructure.com/courses/1094345/pages/resources)** )

2. Fixed detection

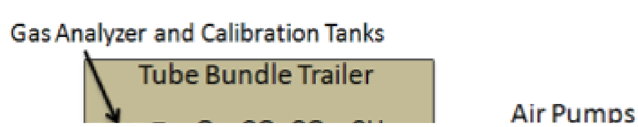
Fixed detection can be telemetry based (e.g., mounted in a specific area) or equipment based, such as machine mounted detectors.  Generally, operators are responsible for monitoring gas concentrations on machine mounted equipment, and the equipment is set to shut down at a threshold value or the operator is trained to shut the equipment down.  For instance, in the US, continuous mining machines will shut down at 1.0% methane.  Miners are trained to withdraw from an area at 2.0% methane.  There is a safety factor included in these levels, as the lower explosive range of methane is 5%.  Telemetry based detection could be set to shut down equipment at a threshold or miners might be notified, for example, if a belt carbon monoxide sensor reaches 50 ppm, a crew working inby the sensor might be instructed to withdraw.

3. Continuous real-time monitoring

Fixed detection generally consists of telemetry based gas sensors (although in some cases they may be wireless).  In the US, they are most often fixed carbon monoxide monitors on conveyor belts, since conveyor belts are acknowledged to have higher relative risk of fire.  There are examples of salt mines in the US that are prone to periodic methane inflow, and fixed telemetry systems will automatically de-energize mine wide (non-emergency) systems or a specific zone of the mine if methane is detected.  In other countries telemetry based methane monitoring may be used more frequently to de-energize zones in a coal mine if methane levels rise.  This system is commonly seen in Australia.  Finally, telemetry based monitoring can be used in a number of ways to achieve efficiency and safety gains via ventilation-on-demand, which is covered in the automation course.  Machine mounted sensors may also (and often are) monitored in real time. Continuous real time monitoring systems are generally monitored in real time.

4. Remote continual monitoring

For all intents and purposes, the continuous real-time monitoring reference above in (3) is real-time, although there may be a delay on the order of milliseconds between the time a concentration is measured and the time the measurement is displayed or recorded.  There could also be several seconds between measurements.  The distinction here between continuous and *continual* is that in the latter there is substantial lag (on the order of several minutes to several hours) between measurements.  The best example of this type of monitoring is a tube-bundle system.  Tube-bundle systems are commonly used to monitor gobs (especially spontaneous combustion prone gobs in Australia) and there are at least two such systems installed in the US in 2017.  Tube-bundle systems physically extract an air sample from a mine via a small diameter tube that may be up to a mile or longer.  The sample is pumped to a central sampling station for analysis.  A single system might have 40 such tubes, and a single sample may require three minutes for analysis.  Samples can be queued in a custom configuration so that data from higher risk points could be collected more often, but it is easy to see how lags develop in such a continual system.  These systems are excellent for monitoring slow trends, such as spontaneous combustion where heating occurs relatively slowly.  They do require substantial maintenance, and the tubes themselves are subject to damage in an explosion, although the advantage is that there is still some access to the atmosphere post emergency.



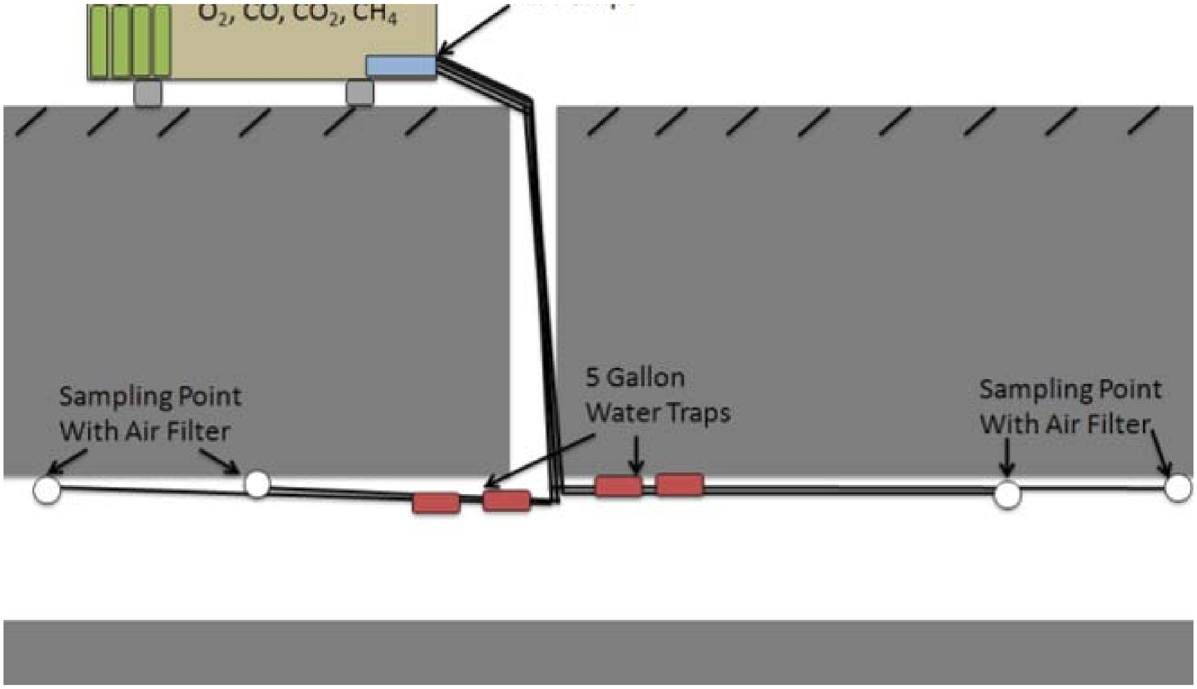



Figure 3.1. Side view schematic of a tube bundle system.

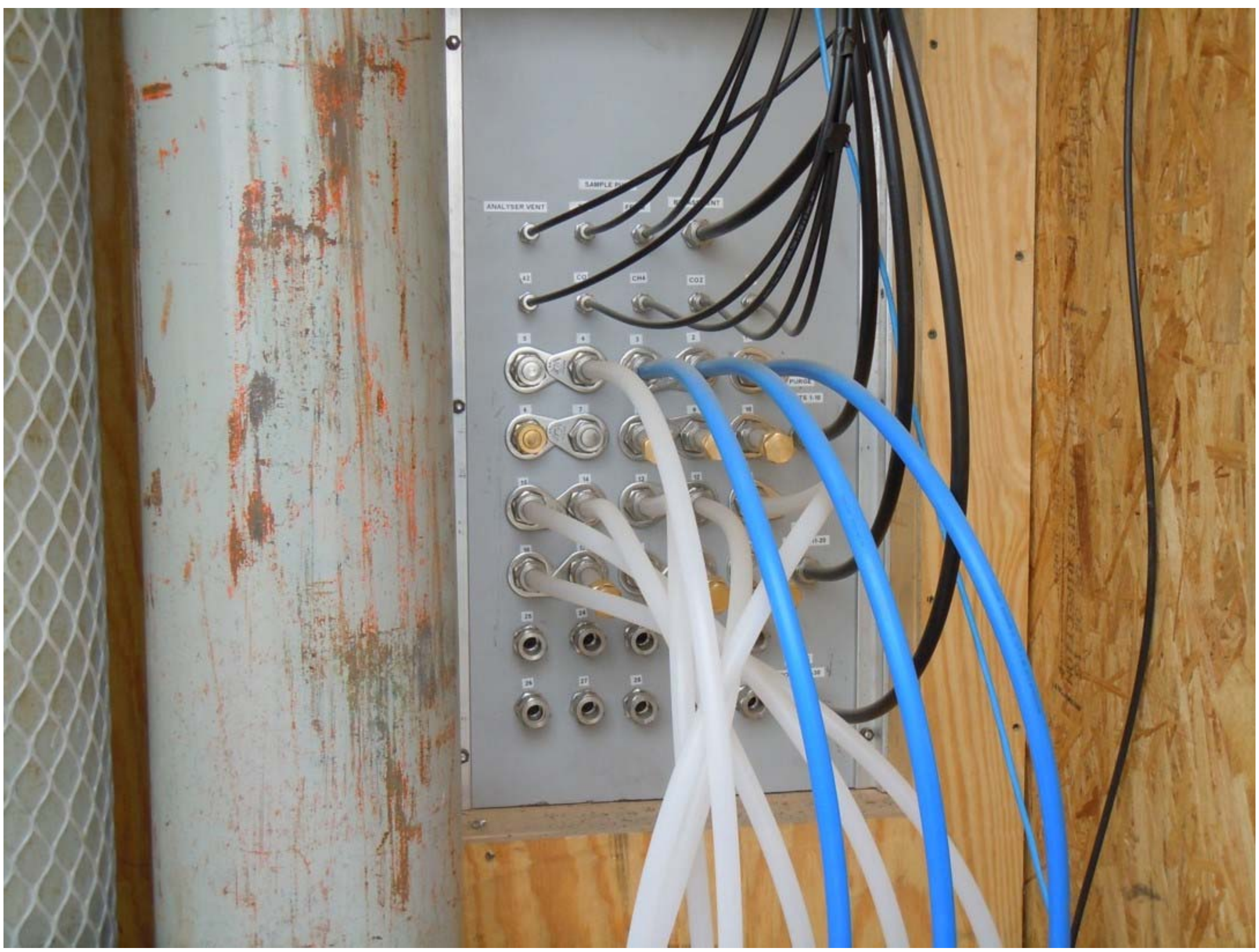


Figure 3.2. Tubes entering a sampling trailer on the surface.

Figure 3.3. Inside of a sampling trailer with analyzer shown in top middle.

A case study involving a tube bundle system at an unnamed underground coal mine in Australia is given below.

**<u>Case Study: Australian Tube Bundle System</u>**

This mine has been monitoring gob via tube bundle since the 1990's, and they have 40 monitoring points with 3 currently idle. The majority of monitoring is at the longwall gate road seals, although they also monitor a few select return points where continuous monitoring is not required. Sampling times are anywhere from 10 minutes to over an hour. The time to sample a specific point is dependent on two variables: first, the distance from the sample point to the sampling trailer, and second, the wait for the infrared (IR) analyzer. The IR analyzer produces sample results in approximately 3 minutes. So, if there are 40 points, it will take at least 2 hours to sample through all of the points. It is likely that there will be points that represent a higher risk area that the ventilation officer will want to see sampled more frequently. He can program the system to analyze that particular sample as often as he likes, for example, every 30 minutes, which will further extend the wait for other samples. This is an excellent way of observing trends, but not a real time system.

The software is set to allow for integration of tube bundle and real time data from other telemetry based systems.

A series of Trigger Action Response Plans (TARP) are developed for most possible atmospheric scenarios that indicate what the plan of action is for specified gas levels. Although a control room operator will call a responsible person if he recognizes that an action level is being approached he has the authority and is expected to follow the TARP plan which can escalate to the evacuation of the mine.

Maintenance includes moving the sampling points and testing the lines for integrity on a monthly basis. Line testing is generally done with compressed air, by hooking an airline to the underground end and checking to see if pressure is maintained at the surface end. Often lines will have several leaks and water is used to identify the leaks, which can be fairly time-consuming. The air tubing costs about the US $0.37/ft.

5. Bag sampling and analysis (by gas chromatography or mass spectrometry)

This type of sampling is generally done on a schedule or during a mine emergency. Samples are either analyzed at a machine on site or sent to a laboratory. Samples may be collected routinely as a way of confirming function of other detection levels or to provide a third party analysis (for instance, to an insurance company), to prove that a mine should still be classified as "non-gassy" or to investigate unusual findings with other detection methods. Additionally, GC is typically utilized during mine emergencies, particularly those involving fire and explosion in order to ascertain current conditions underground as well as the development of combustion.  In fact, in the US, MSHA maintains multiple portable systems as part of their emergency response units.


Figure 3.4. MSHA's mobile gas laboratory ([LDV, 2017. (https://canvas.instructure.com/courses/1094345/pages/resources)](https://canvas.instructure.com/courses/1094345/pages/resources) )

# Course Syllabus

Jump to Today ✎ Edit

**Ventilation Simulation**

**Predicting conditions for miners**

Mining professionals must know the quantity, quality, and cost of air delivered to the working areas in an underground mine. The conditions are always changing as mining progresses and a prediction tool must be used to correlate measurements taken in ventilation surveys with future mining conditions and locations. This module introduces the concepts behind ventilation simulations and gives a platform for software vendors to deliver or to link to material specific to their solutions.

**Learning Objectives**

1. **Describe simulation purposes, processes, and limitations**
2. **Create a simulation**
3. **Conduct analysis and equivalence calculations**

**Get Started**

**Go to the Course Index**

**About the Aeolus Project**

# Course Summary:

| Date | Details |
|---|---|

# 1.0 Introduction

Although the practice of providing ventilation for underground mines has existed for centuries, the simulation of underground ventilation systems is much more recent. The first computer-aided ventilation network simulators emerged in the 1970's, shortly after the popularization of "computers" as means to solve complex, or redundant mathematical problems.

Before that time, ventilation knowledge was gathered empirically through years of experience, and wholly unable to be transferred or replicated outside of the originator. Ventilation planning at that time, was an iterative practice, dogged by uncertainty, and based solely upon past observations and results. Perhaps rightly so, mine ventilation was seen as a "dark art" and its practitioners were often mistrusted.

Today, powerful mine ventilation network simulators provide knowledgeable ventilation engineers and technicians with the tools to manage and simulate large and complex mines with relative precision and accuracy. In fact, an accurate ventilation model has become so vital to the success and viability of modern underground mines, that few operate without them.

In this module, we will explore the fundamental processes and strategy behind constructing, maintaining and expanding a ventilation model, while avoiding the specific commands, operations and actions that are unique to individual software packages. Detailed descriptions of the numerical and analytical methods for solving complex networks will also be avoided, as these processes are now completely internal to ventilation network simulation programs, and are not required in order to successfully create and manipulate the ventilation models.

Complex mine ventilation networks, or “systems” are indeed just that- systems. It is important to consider that they function both as a system with defined boundaries (e.g., tunnel walls, fans, surface connections, etc.) and as part of a larger system of mine operations that includes production, development, maintenance, transport, geomechanics and other activities that all influence each other. This “systems approach” to ventilation will be emphasized throughout this module, as it is critical to the design and simulation of a successful mine ventilation system.

# 2.0 Fundamentals of Network Simulation

Before the process of building a ventilation model is begun, several questions must be answered. These critical questions include:

- What is the purpose of the ventilation model?
- What are the model design criteria (parameters of model construction and evaluation)?
- What information is desired from the outcome, and at what level of detail?
- What software package will be used to perform the simulation(s)?

The answers to these questions are unique to each ventilation model and simulation. For example, is the model designed to provide the basis for a major ventilation upgrade at an existing mine, or establish the feasibility of a potential mining project? Will the mine conform to norms in the country where its located, or the country of the company's headquarters, or both? Will the results be used to determine is the mine has sufficient airflow for the equipment, or will it be used to make decisions regarding large capital expenditures (often worth millions of dollars)?

Obviously, there is a great deal of variation in the purpose, level of detail and application in ventilation simulations, and these differences will have profound influences on the ventilation models themselves. Understanding the differences between conceptual-level studies and detailed engineering studies is critical to obtaining accurate outcomes and avoiding costly errors.

# 2.1 Models of Planned or Future Networks

If no physical mine or network exists, such as in the simulation of mine ventilation systems as part of a conceptual design or “feasibility study”, the process for constructing the ventilation model relies extensively on assumptions regarding the theoretical system.  These assumptions include not only the physical parameters of the mine network (e.g., locations, dimensions and resistance of the mine tunnels and shafts) but also many environmental parameters such as elevation, air temperature and relative humidity (often expressed as a range of values).

# 2.2 Models of Existing Networks

Models of existing mine ventilation systems should be based on the actual, measured conditions in that mine as closely as practicable. Although it is often not possible to measure the resistance in all existing flow paths due to the size or complexity of the mine, or other restrictions on access, measured values of resistance should be utilized whenever possible. Actual resistances for flow paths within the mine may be determined from concurrent differential pressure and airflow quantity measurements (via the Square Law) or from Atkinson friction factors (k-factors) that were measured underground in representative airways.

Ventilation controls that act as airflow regulators should be measured directly and entered into the ventilation model from a measured differential pressure drop and airflow quantity. Operating fans should be simulated according to their characteristic operating curve(s) (provided by the manufacturer) when corrected for actual air density, blade pitch and operating frequency or rotational speed as appropriate. Most modern ventilation network simulation programs will automatically correct for fan curve density, and frequency provided that the appropriate characteristic curve is selected.

When simulating an existing mine ventilation network, care should be taken to replicate the existing mine exactly, and not to model the ideal or desired conditions where those differ from the actual circumstances encountered in the mine. For example, if doors in the main decline of an underground ramp are intended to stay closed, but in practice are left open to facilitate traffic, then these doors should be simulated in the open position for any ventilation model of the current ventilation system. Failure to simulate actual conditions where they differ from the theoretical or ideal ventilation system design is one of the most common errors committed by novice ventilation system modelers.

The actual performance of any mine ventilation system may be determined from a mine ventilation survey, or audit. Additional information about the planning, execution and evaluation of mine ventilation system performance may be found in the “Ventilation Surveys” module of the Aeolus Project courses.

# 2.3 Compressible Flow Models vs. Incompressible Flow Models

Air, a mixture of constituent gases with a generally known content flows through mine circuits as a compressible fluid. Unlike water, for example, the density of mine air can vary significantly throughout the mine circuit as a result of changes in elevation, temperature, humidity, barometric pressure and due to the influence of any fans present in the circuit.

Although most mine ventilation design criteria and regulations are specifically concerned with airflow volumes, these will change throughout the mine, even in entries where no air enters or leaves (e.g., slopes or shafts). It is important to remember that although these volumes can change based on the local air density at the various points of measurement, the Law of Conservation of Mass is observed throughout the circuit. This means that the mass flow of air through the mine remains constant. This distinction will be important to understanding mine airflow circuits, and performing accurate mine ventilation simulations.

In some cases, the simulation of compressible flow-networks such as mine ventilation systems can be done using incompressible-flow simulation programs (i.e., VnetPC, I-Camps) provided that certain criteria are met. If the mine network lies in a relatively flat plane, and the fan pressures encountered are low, incompressible-flow simulators can provide sufficiently accurate results for simulation; however, in deep mines, mines with significant changes in elevation, or simulations that must account for significant heat additions (e.g., from mine equipment or rock strata), additional work will need to be performed to account for the changes in air density that will occur within the circuit. This may include the use of "injection" or "rejection" branches that add or remove airflow volume from the circuit to approximate the change in airflow volume resulting from the density changes. Additional care should always be taken when specifying fans that the correct air density for the fan installation has been considered when using incompressible flow programs.

With the evolution of mine ventilation science and the vast increase in computing power realized over the past several decades, compressible-flow network simulation programs such as Ventsim and VUMA now allow the simulation of mine ventilation networks to account for changes in airflow density to be computed automatically throughout the mine circuit provided that accurate input data (e.g., mine environmental data, heat flow, etc.) is provided. These programs are especially well-suited to deep mine network simulations, simulations of heat-flow and the modeling of underground fire scenarios.

Whatever software package is being used, it is critically important to understand how it simulates air flow (whether it assumes incompressibility or whether it simulates air as truly compressible) and what input parameters are required in order to obtain accurate results.

# 2.4 Ventilation Model Design Criteria

The design of a ventilation system for a mine is a process that must account for a wide range of parameters that affect the mine ventilation, as well as the intended demands on the system. Some of these factors included the physical location of the mine workings as well as the geography and climate of the area. Local government regulations and policies are other parameters that dictate the size and scope of mine ventilation systems. Ultimately, the decision of which design criteria to use will have a profound impact on the results of the simulation.

In this section, we will discuss some of the ventilation system design criteria that may be encountered or utilized in the process of model construction. Additional Information regarding project design criteria may be found in the "Ventilation Surveys" and "Metal and Non-Metal Mine Ventilation" courses that are part of the Aeolus Project.

The total airflow requirements for most mines are based on the number and type of equipment expected to be in use and the number of people working underground. In general, the mine will be ventilated sufficiently to dilute the exhaust gases and particulate emissions of the Diesel equipment in use. Personnel airflow requirements are utilized to calculate the required airflow for shops, refuge stations, and other areas where large groups of people may congregate. Occasionally, airflow volume may be calculated based on other metrics such as airflow velocity, “air changes” or the need to dissipate heat, such as in large electrical caverns or motor rooms.

The total airflow quantity required should be matched to the cross-sectional area of the mine airways in order to maintain an acceptable velocity. Airflow velocities should be maintained within proscribed limits that are designed to ensure the safety of persons working or traveling in the underground. A balance is required, whereby the ventilation system provides sufficient flow to remove contaminants from the working areas, while avoiding generating and entraining dust and potential discomfort caused by high velocities.

Although maximum, not to exceed upper velocity limits are often given, it is generally not optimal to utilize these values as part of the design. Likewise, although a minimum acceptable airflow velocity is recommended for most mining applications, this value should not be used as or assumed to be a target value.

There are several potentially hazardous gases and gas mixtures that can occur in underground mines and tunnels. These gases/contaminants can be the product of strata gas inflows, emissions from mine equipment, blasting fumes or due to chemical reactions (e.g. oxidation of sulphide ores).

Additional factors that may negatively affect the underground mine environment include mineral dust(s) and Diesel Particulate Matter (DPM) or “respirable combustible dust” (for the purposes of this course, the terms “Diesel Particulate Matter”, “DPM”, and “respirable combustible dust” are considered equivalent and are used interchangeably).

Many toxic gases can be found in underground mines, whether naturally occurring or as the result of normal mining activities (blasting, haulage, etc.). Criteria for the exposure to toxic substances is often governed by local regulations, which should be considered the minimally acceptable limits for any regulated substance. Often, organizations such as the American Conference of Industrial Hygienists (ACGIH), the World Health Organization (WHO), the International Agency for Research of Cancer (IARC) publish additional guidelines for noxious gases, dusts, etc. In many locations, the duty of care requires compliance with the ALARA/ALARP principal, which requires that all risks associated with a particular threat to human health be reduced "As low as reasonably achievable/possible".

Environmental parameters such as elevation, air temperature, relative humidity will all have an effect on the simulation and the resistance of the airflow circuit(s). Depending on the simulation and the context, other environmental parameters such as Virgin Rock Temperature (VRT), Geothermal Gradient, Rock Thermal Diffusivity and Conductivity may also be required.

Finally, resistance values for tunnels, shafts and ventilation controls (e.g., walls, doors, regulators, etc.) can be obtained from a variety of sources. These could include fixed resistances representative of "typical" mine installations, k-factors that were obtained from general lists or measured values, or direct measurements of pressure and quantity through a specific section of tunnel or shaft.

Physical parameters for the simulation, including tunnel locations, elevations, cross-sections, etc., can be approximated, or more often obtained directly from the various mine design software packages. Often, these physical parameters can be imported directly into the ventilation simulation package.

There are many ventilation network simulation software packages available for use. Some of these include Ventsim, VnetPC, VUMA, I-Camps and Multiflux. These programs are all different, have different strengths and weaknesses, as well as specific applications where they excel. Examples used in this module will given using Ventsim. Ventsim is a popular, and powerful ventilation network simulation program, which freely provides its User's Manual and a file viewer as well as many helpful training tools.

# 3.0 Model Construction

The exact processes and commands that must be used to construct ventilation models are somewhat unique to each individual software package. For that reason, the instructions for modeling will be specific to the software package used. In this case, the selected software package is Ventsim, the most popular and widely utilized ventilation network simulation software currently available.

The User's Manual for the Ventsim ventilation network simulation software package is provided below. It is a comprehensive guide to the use of the software for almost all applications likely to be encountered by a novice user.

**Ventsim User's Manual (https://canvas.instructure.com/courses/1175699/files/69802545/download?wrap=1)**

Much of the information that is given will be relevant to modeling in any software package, and if additional, software-specific resources are needed, please contact the software provider directly to obtain assistance.

# 3.1 Building a Network from Scratch

Although it is more common to replicate mine networks from existing mine design files, in some cases it may be necessary to draw an approximation of a mine ventilation network free-form. It also gives several helpful basic tips required for model construction. Although simple in nature, the processes and functions demonstrated may be repeated to complete other, more complex models and simulations.

The following tutorial shows how to construct a basic ventilation network in Ventsim.

**Building a Mine Network from Scratch in Ventsim** (http://www.ventsim.com/tutorials/tutorial-creating-a-basic-model-video/)

# 3.2 Building a Network from an Existing Mine Model

In most cases, mine ventilation simulations will begin with a physical mine network developed from measured survey data or from planned development designed through one of the leading mine planning software packages (SurvCAD, Vulcan, Mine24D, Deswick, etc.).

The exact steps required to import a physical mine design to a ventilation network simulation program will vary based on the program used.  The model development process in Ventsim is shown in the following video.

**Importing a Mine Network Into Ventsim (http://www.ventsim.com/tutorials/tutorial-creating-model-cad-file-video/)**

# 4.0 Fans and Other Ventilation Controls

What makes air flow through the mine, or mining circuit? In order to air to flow, or move from one location to the other, a pressure differential must exist. In some cases, this pressure differential is the result of differences in air density between the two locations. This is often referred to as Natural Ventilation Pressure, or NVP. The magnitude of the NVP depends on the disparity between the two air densities (at the start and end of the branch or circuit in question).

For most mines, however; the answer to the above question is **Fans (https://canvas.instructure.com/courses/1049400)**. Fans provide the pressure differential to move air in a controlled manner throughout the mine circuit. In most cases, the pressure imparted by the mechanical ventilation (fans) is much more significant to the performance of the ventilation system than the NVP acting on the circuit.

# 4.1 Fans

Each mine ventilation simulation software will have the capacity to enter and simulate the performance of a variety of fans as part of the ventilation model.

In Ventsim, fans are included in the ventilation model by clicking on the fan location using the edit tool and selecting the fan tab, or by clicking on the branch using the fan tool.  This opens the fan dialogue box, as shown in Figure 4.1.1.

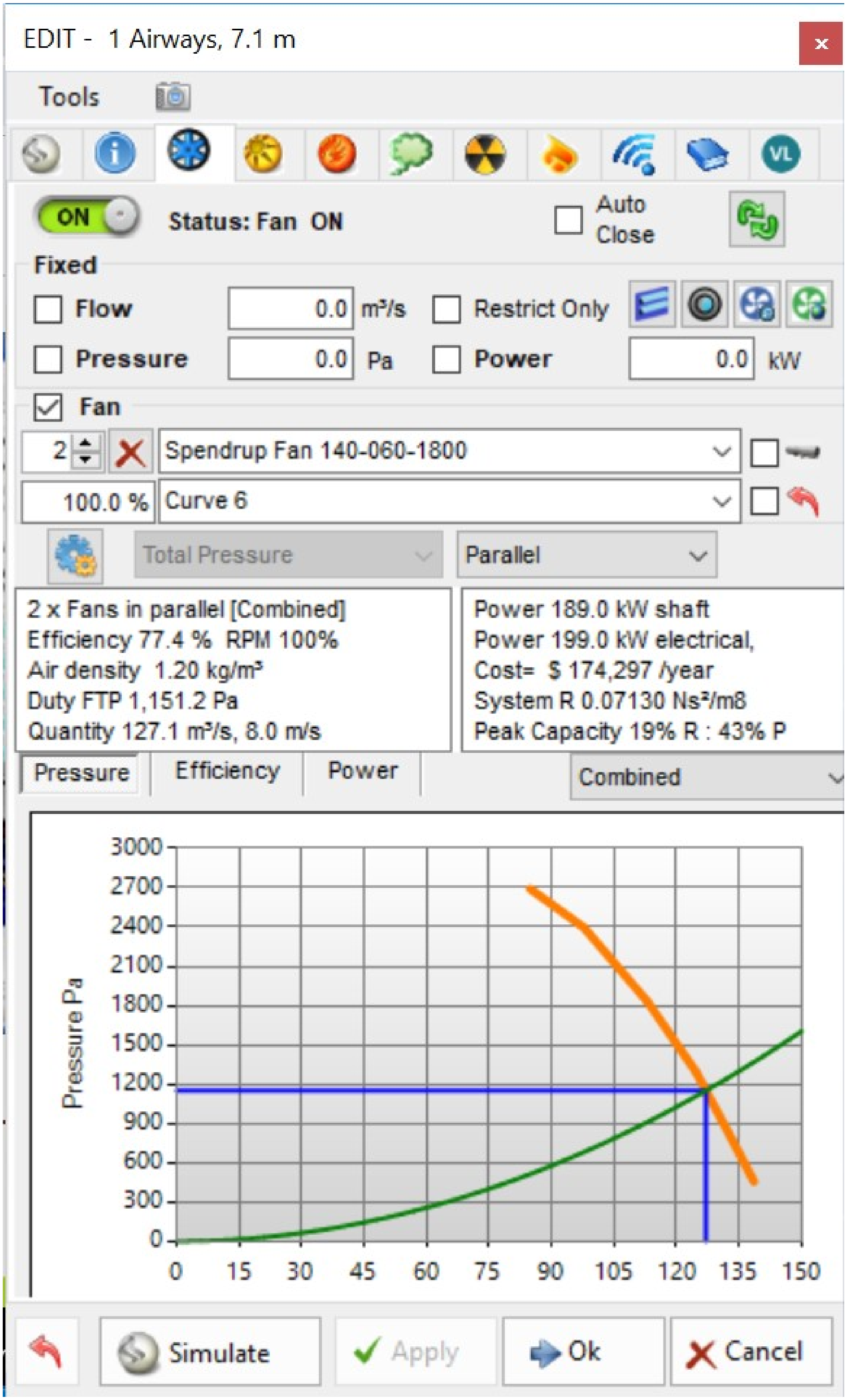



**Figure 4.1.1. Fan Dialogue Box in Ventsim.**

From this screen, the user can specify the number and configuration of fans in the installation. By clicking on the edit button (gear icon), it is possible to enter the manufacturer's fan curve directly into the model. This is shown on Figure 4.1.2.

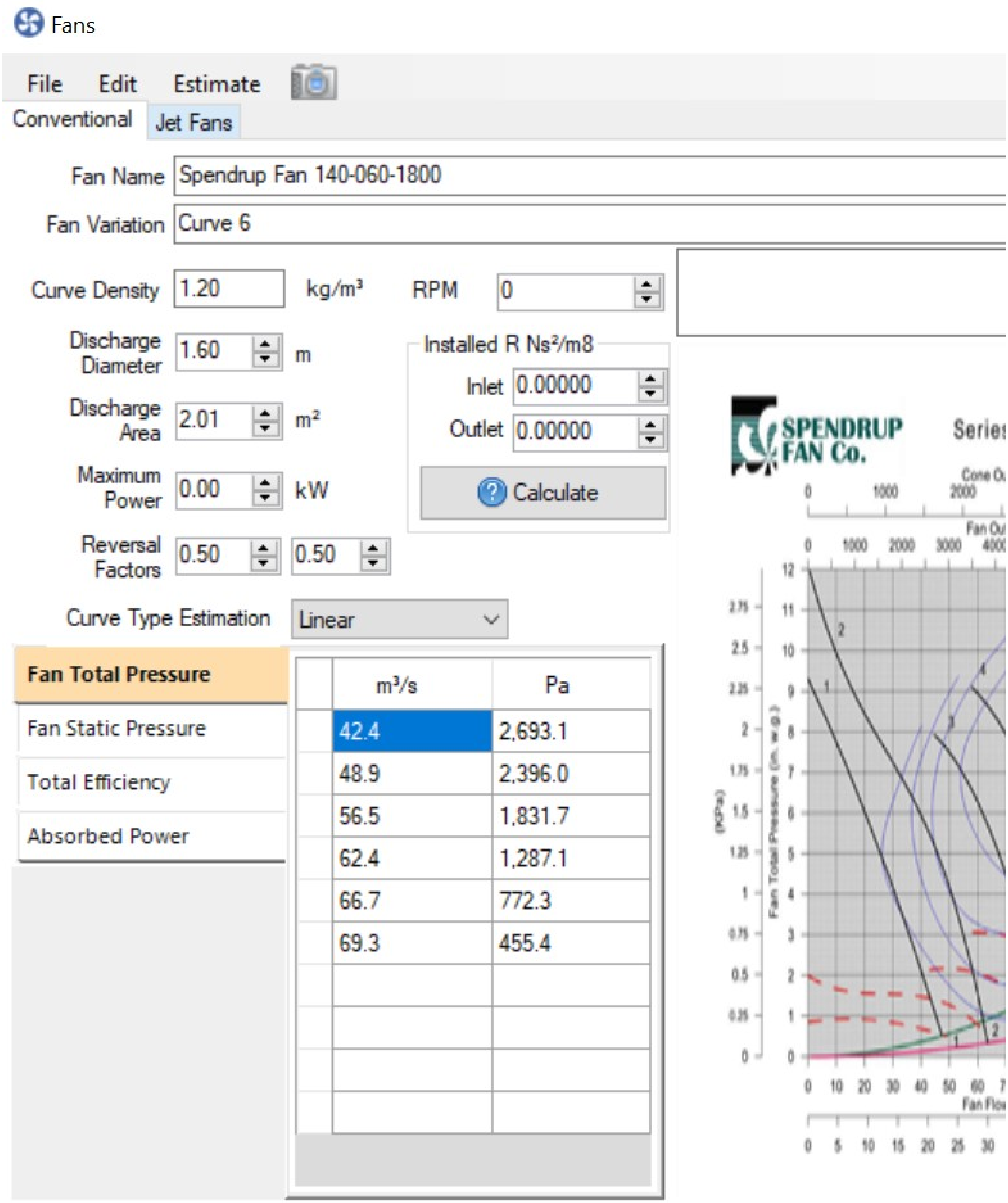


**Figure 4.1.2. Fan Curve Dialogue Box in Ventsim.**

Using Ventsim, it is possible to "drag and drop" a digital image of a fan curve file directly into the fan digitizer, and automatically generate the fan curve, taking care to specify the curve density and rpm or operating frequency. Fan curves can also be entered manually through the table provided. Once a fan curve has been entered, it will be possible to change the frequency of the fan input power to simulate a Variable Frequency Drive. Ventsim will automatically determine the calculated air density for the fan installation if the model is operating in compressible-flow mode.

Note that if the fan curve or motor does not support the operating point determined by the simulator, the fan will enter into "stall", an unstable condition that can result in decreased performance or, in extreme cases, fan failure and destruction.

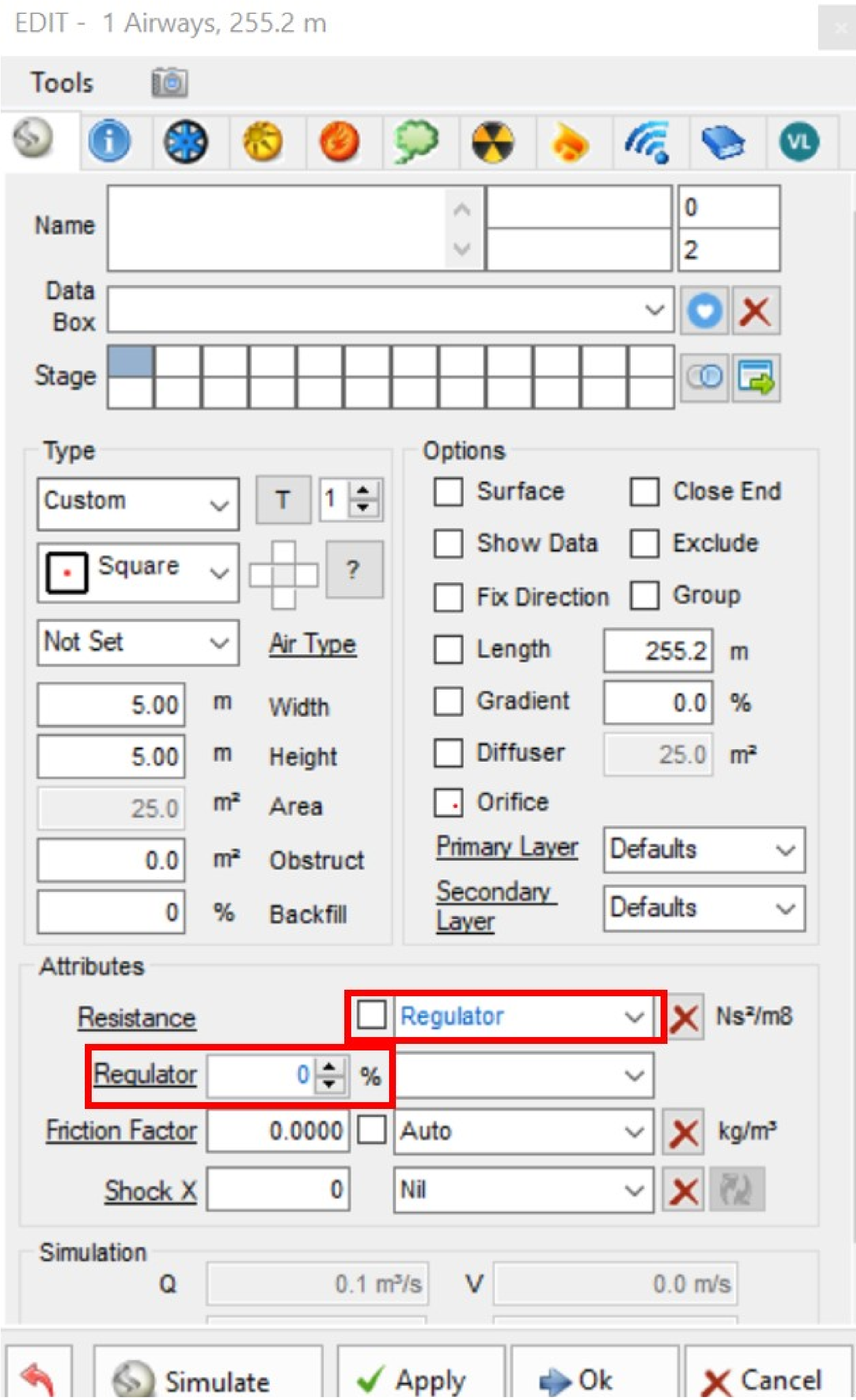
Note that the Fan Dialogue Box also allows the user to input a "Fixed Quantity" or a "Fixed Pressure" into a branch. In the case of a Fixed Quantity, the model will force the selected quantity through the branch regardless of the conditions or pressure differential required.

When a Fixed Pressure is selected, the model fixes the pressure constant and varies the airflow volume according to the mine conditions determined by the simulator. These commands are used when a fan curve is not available, or in some cases when predicting future networks, when a certain required quantity is desired, and to determine whether a regulator or booster fan will be required. Fixed Pressures may also be used in some cases to approximate the affect(s) of NVP on the mine ventilation network, particularly when incompressible-flow models are used.

Care should always be taken when utilizing these features, as they can often lead to erroneous results if not used appropriately. In fact, the overuse of fixed quantities is one of the most common errors encountered in ventilation models constructed by novice users.

# 4.2 Regulators

Flow regulators in the mine can be simulated in several ways. By using the Edit tool and clicking on the selected location, a resistance for the regulator can be entered. If the resistance has been measured, it can be entered as a pressure and quantity. Alternatively it can be entered by selecting "Regulator" and entering the percent open. This is shown in Figure 4.2.1.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Show Data
Exclude
Fix Direction
Group
Length 255.2 m
Gradient 0.0 %
Diffuser 25.0 m²
Orifice
Primary Layer Defaults
Secondary Layer Defaults
Attributes
Resistance
Regulator
Ns²/m8
Regulator 0 %
Friction Factor 0.0000
Auto
kg/m³
Shock X 0
Nil
Simulation
Q 0.1 m³/s
V 0.0 m/s
Simulate
Apply
Ok
Cancel

**Figure 4.2.1: Regulator Resistance - % Open.**

Alternatively, the Orifice option can be selected, and the area of the opening can be entered if it is known (or can reasonably be estimated). This is shown in Figure 4.2.2.

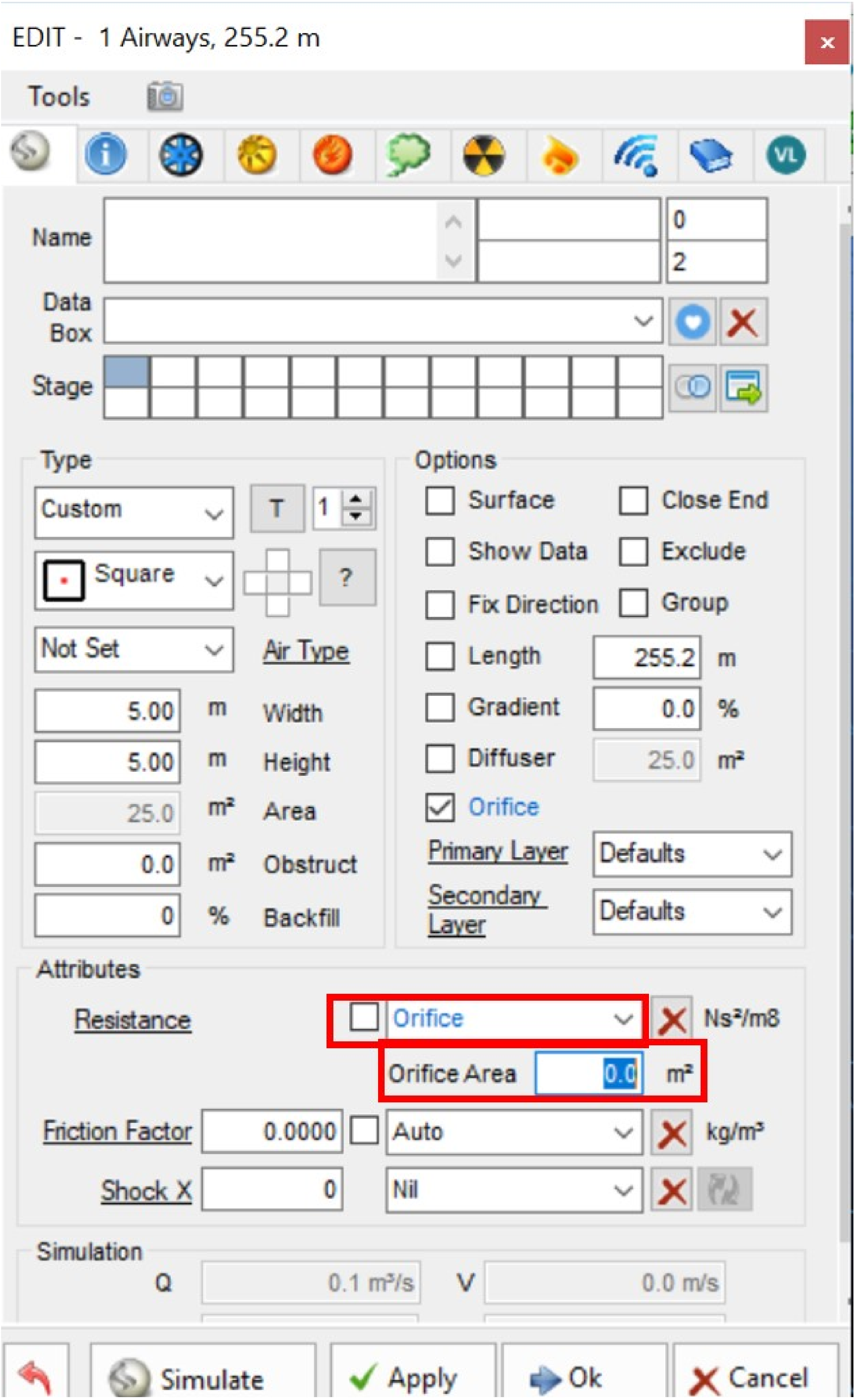

EDIT - 1 Airways, 255.2 m
Tools
VL
Name
0
2
Data Box
Stage
Type
Custom
T
1
Square
?
Not Set
Air Type
5.00 m Width
5.00 m Height
25.0 m² Area
0.0 m² Obstruct
0 % Backfill
Options
Surface
Close End
Show Data
Exclude
Fix Direction
Group
Length 255.2 m
Gradient 0.0 %
Diffuser 25.0 m²
Orifice
Primary Layer Defaults
Secondary Layer Defaults
Attributes
Resistance
Orifice
Ns²/m8
Orifice Area 0.0 m²
Friction Factor 0.0000
Auto
kg/m³
Shock X 0
Nil
Simulation
Q 0.1 m³/s
V 0.0 m/s
Simulate
Apply
Ok
Cancel

**Figure 4.2.2: Regulator Orifice - Area Open.**

In some cases, where a future scenario is being modeled, a Fixed Quantity may be used to determine if a regulator is needed, and approximately how large of an orifice (opening) it should provide. Care should always be taken when using Fixed Quantities, and these should be replaced with "Orifices" or "Regulators" within Ventsim as soon as reasonably practicable.

# 4.3 Fixed Resistance Controls

Other ventilation controls with fixed resistances (e.g., walls, doors, curtains, airlocks, etc.) are also helpful tools in simulating mine ventilation circuits. All ventilation network simulation software packages will offer some means of entering a fixed resistance into a network branch in order to simulate the resistance of various ventilation controls.

In Ventsim, several options are included in the resistance dropdown menu of the EDIT box. Additional, customizable options can be entered in by selecting "Settings" and clicking on "pre-sets" and "Resistances". From here it is possible to enter any number of ventilation controls and their associated resistance values, that will then be accessible from the Resistance dropdown menu in the EDIT box.

Actual resistance values for ventilation controls can be obtained by measuring controls in an existing mine, and representative values can be obtained from published values in papers and textbooks.

# 5.0 Model Validation

Validation of existing mine network simulations is an important part of the simulation process. If the mine ventilation model of an existing mine does not sufficiently mimic, or predict the behavior of the ventilation system, then its value is questionable.

Without an accurate ventilation model, the degree of confidence in any planning or trouble-shooting exercises will be low, and any conclusions drawn from the simulations results is likely to be in error.

# 5.1 Establishing Model Accuracy

Although ventilation software offers "mathematical perfection" when solving complex mine networks, the reality is that error exists, both in measured data, and in the approximations of mine resistance and assumptions upon which our models are built.  It should never be assumed that a ventilation model, no matter how carefully constructed, represents a thoroughly accurate approximation of a mine network without verification.

Just how accurately should a ventilation model predict the actual conditions in a mine or underground circuit?  That question likely depends on the user, and the application, but generally speaking, any model that predicts the flow quantities in the mine circuit with 90% Accuracy / 10% Error is considered acceptable.

A mathematical approximation of the models accuracy may be obtained by dividing the sum of the differences between measured airflow quantities and predicted airflow quantities by the sum of the total measured airflow quantities.  This will provide a "correlation error" usually reported as a percent (%) error.  It may be expressed in terms of model accuracy by subtracting the correlation error from 100%.

Ventsim creators Chasm Consulting offer the following suggestions for creating accurate ventilation models:

1. Where airway sizes or resistance cannot be accurately measured, use the mine design to help create the Ventsim™ model, and use the actual survey data to improve the Ventsim™ model airway size estimation. Most mines will commonly overbreak the design size so that final airway size may be 10 – 15% larger. If this is not accommodated in the Ventsim™ model, the model may over predict the pressure or under predict the airflow.
2. Don’t forget to consider Shock Losses, particularly at major intersection and junctions with lots of airflow. Shock Losses can commonly add 10 – 15% additional pressure requirements to a fan and must be considered. Funnily enough, engineers who do not consider over-break from design sizes, also forget to consider shock losses, and the effect somewhat cancel each other out across the model – but not always in the correct places.
3. If you cannot perform a pressure / resistance survey across the mine, then at least try to measure some example resistances in main airways. These can be used to derive friction factors which can then be applied to other similar airways, and are probably better than using standard default friction factors.
4. Ensure all simulation parameters and settings are accurate. This step requires a systematic audit or all information used and will be covered in more detail in a future article. Each major ventilation assumption (resistance, friction factor, airway size, fan curve, shock losses, simulation settings such as compressible flows and surface temperatures and barometric pressures) needs to be reviewed for accuracy. Most Ventsim™ data can be displayed in different colours so it is easy to examine an entire model for colours that are not consistent with what you would expect.

5. If the mine is known to have a strong natural ventilation pressure presence, then this will need to be included in the model. While the Ventsim™ automatic natural ventilation option can be used, unless a very accurate heat simulation model is made of the mine this will not give the correct result. In this case, it may be best to TURN OFF automatic natural ventilation, and use FIXED PRESSURES in the surface airways to simulate known natural ventilation pressures.

Although it may be tempting to blame the software for errors encountered in the simulation process, the reality is that simulation errors are rarely the fault of the software, and almost always the result of errors made in the selection or input of data by the user.

# Underground Coal Mine Ventilation Design

Jump to Today Edit

**Coal Mine Ventilation Design**

Coal mines are some of the most heavily regulated underground mines from a ventilation standpoint. International approaches to safety and regulation vary substantially. For instance, in some countries the use of belt air and underground main mine fans are common. This module is developed from the US perspective where such practices are uncommon or heavily regulated.

The AEOLUS Project

**Learning Objectives**

1. **List parameters that influence ventilation system design.**
2. **Demonstrate how to validate a ventilation model for use in ventilation system design.**
3. **Explain the difference between geologic, environmental and strategic design factors.**
4. **Identify various strategies for auxiliary ventilation.**
5. **Explain the mechanics of how airflow is used to dilute and remove contaminants.**
6. **Demonstrate how to select fans utilized in underground mine ventilation systems.**

Get Started

Go to the Course Index

About Aeolus

## Course Summary:

| Date | Details |
| --- | --- |

# 1.0 Basis of Design

Prior to beginning any ventilation design project, it is useful to ask “what is the point?” Establishing the purpose of your study can help you define both the start and end of the project, as well as what path you will take along the way. Defining the Basis of Design (BOD) is more than simply deciding on the design criteria (although that is an important part of the exercise). The reasons for conducting a ventilation design may vary greatly, from a simple “what if” exercise, conceptual design or detailed engineering study. The required inputs, and level of detail will also change proportionally. In the US, the primary issues related to ventilation of underground coal mines are dilution of dust to protect miner health and reduce explosives risks, dilution or removal of methane to reduce explosive risks, and adequate ventilation of all areas to ensure that low oxygen is not an issue.

# 1.1 Introduction

*Much of this section is reproduced under Introduction to Metal/Nonmetal Ventilation Design, but gives an important overview of design principles.*

As ventilation modeling tools (i.e. software packages) have evolved and become more powerful and user-friendly, a worrying consequence has also manifested itself in the practice of mine planning professionals. Although current iterations of popular mine planning software packages (e.g., Ventsim, VUMA and VnetPC) have all been proven reliable and useful planning tools, it should be remembered that their output is only as good as the inputs provided. Further to this point, it is also possible to misuse, or misinterpret even reliably accurate results if the user is ignorant, or inexperienced with regard to ventilation practice.

Coal mine entries tend to be developed and retreated quite quickly in comparison to many metal and non metal mines so it is also critical that models be updated and calibrated regularly. Many professionals update and calibrate at on at least a quarterly basis.

Dr. Rick Brake, founder of Mine Ventilation Australia has identified the following areas in which mistakes of this nature are most likely to occur:

- Failure of the principals to understand the project scope, and expected outcome(s) or deliverables.
- Failure of the study authors (designers) to use the appropriate inputs, assumptions and/or design criteria.
- Failure to develop or use a correct or *validated* ventilation model.

Dr. Brake goes on to surmise that although inexperienced ventilation practitioners are often guilty of making one (or more of these mistakes), these failures are not limited to just those ventilation engineers and technicians that are inexperienced or new to the subject matter. Even experienced, knowledgeable staff can make mistakes, if the impacts of various design assumptions or operating practices are not well considered or understood. In light of the significant negative impacts to health and safety and project economics that can occur when incorrect ventilation designs are implemented, it is critical for ventilation planners to adopt the following standards of practice with respect to ventilation system designs:

- Always use a properly correlated, or validated ventilation model as the basis for performing and predictive design.
- Always prepare an appropriate BOD report prior to beginning a ventilation project or study.

It is important to note that a properly validate model does not necessarily indicate a properly ventilated mine, or even a properly designed one. In fact, this is simply an indication of how accurately the ventilation model predicts the behavior of the actual ventilation system.

More detailed information about constructing and validating ventilation models may be found in the “Ventilation Modeling and Network Simulation” course attached to this series.

It is unfortunately not uncommon for a well-correlated ventilation model to accurately predict the behavior of a very poor ventilation system and/or design. Thus, in order to ensure the successful completion of any modeling exercise, the BOD must be effectively executed.

Ultimately, the success of the study or project will depend upon the following critical criteria:

1. A well-defined and understood scope of work including all desired outcomes and deliverables.
2. A properly validated ventilation model.
3. A properly constructed and documented BOD.
4. Execution by competent, experienced ventilation professional(s).

When the above criteria are all satisfied, then the probability of successful outcome for the ventilation study has been maximized.

Click on the link below to download Dr. Brake's paper.

**QA_Ventilation_Model_Design_Brake.pdf (https://canvas.instructure.com/courses/1049599/files/46434948/download?wrap=1) (https://canvas.instructure.com/courses/1049599/files/46434948/download?wrap=1)**

# 2.0 Environmental Design Issues

In the United States it is unusual to see an underground coal mine deeper than 3,000 feet, underground coal mines in the US can be found in artic and hot climates. Geothermal issues are only rarely encountered.

# 2.1 Regulatory Design Factors

Another critical impact on the mine ventilation system based on its physical (geographic) location is the determination of regulatory oversight for that mine and ventilation system.

The regulations that govern mines, and mine ventilation in particular will vary widely based on the country in which that mine is located, and in some cases, the state or province within that country. In the US, coal mine ventilation is regulated under 30 CFR, and the primary sections for underground coal are:

**Part 70: Mandatory Health Standards for Underground Coal Mines** (https://www.ecfr.gov/cgi-bin/text-idx?SID=444b96b7f19380eacf4023dd11893c1a&mc=true&node=pt30.1.70&rgn=div5)

**Part 75: Mandatory Safety Standards for Underground Coal Mines** (https://www.ecfr.gov/cgi-bin/text-idx?SID=444b96b7f19380eacf4023dd11893c1a&mc=true&node=pt30.1.75&rgn=div5)

**Part 90: Mandatory Health Standards - Coal Miners Who Have Evidence of the Development of Pneumoconiosis.** (https://www.ecfr.gov/cgi-bin/text-idx?SID=444b96b7f19380eacf4023dd11893c1a&mc=true&node=pt30.1.90&rgn=div5)

Additionally, **Part 62: Occupational Noise Exposure** (https://www.ecfr.gov/cgi-bin/text-idx?SID=444b96b7f19380eacf4023dd11893c1a&mc=true&node=pt30.1.62&rgn=div5) , often falls under the purvey of the ventilation engineer, but will not be discussed further in this section.

There has recently been evidence of an **increase** (https://ehp.niehs.nih.gov/doi/pdf/10.1289/ehp.124-A13) in Coal Workers' Pneumoconiosis (CWP), or Black Lung, particularly among Appalachian Coal Miners.

The current **Permissible Exposure Limit (PEL)** (https://www.ecfr.gov/cgi-bin/text-idx?SID=444b96b7f19380eacf4023dd11893c1a&mc=true&node=se30.1.70_1100&rgn=div8) for coal miners in the US is an average of 2.0 mg/m$^3$ during each shift in active workings, one of the lowest standards in the world. However, it is clear that American ventilation engineers have more work to do in protecting the health of coal miners. A recent study by the National Academies of Science sheds more light on the issue - download **Monitoring and Sampling Approaches to Assess Underground Coal Mine Dust Exposures.** (https://www.nap.edu/catalog/25111/monitoring-and-sampling-approaches-to-assess-underground-coal-mine-dust-exposures)

Some examples of ventilation system regulations include, but are not limited to:

- Location of primary fans
- Frequency of monitoring (gas, respirable dust, inert dutt)
- Storage and transport of flammable materials
- minimum air velocity and quantity
- Contaminant exposure

A thorough understanding of all applicable laws and regulations is a required procedure when designing any ventilation system for new or existing mines.

# 2.2 Geologic Considerations

While there is a tendency to focus on environmental factors that exist above the surface at any particular mineral property, it is important to consider that the conditions below the ground will have an equally significant impact on a subterranean mine.

Exploration drilling will reveal details of the mineralization, strata temperatures and morphology of the orebody and host rock that can all affect the underground environment negatively and require mitigation. In this manner, the ventilation system design may depend, at least partially, on the vagaries of geology.

Exploration drilling prior to mine development and during development can provide critical data on methane content and roof and floor conditions.  For instance, shale can be particularly vulnerable to fluctuations in temperature and humidity, and example of the interplay between ventilation and ground control.  Massive fractured sandstone in the roof pose obvious ground control hazards, but it can also pose health hazards, as the regular cutting of sandstone will cause more sparking (explosion hazard) as well as the liberation of more respirable silica dust which has been identified as contributing to rapid development lung disease (silicosis) and has been identified as a **lung carcinogen (https://www.osha.gov/OshDoc/data_General_Facts/crystalline-factsheet.pdf)** .

Finally, in underground coal mines the entry width and height, as well as number of entries and pillar size will significantly impact resistance of the ventilation system which directly impacts operational cost.  The figure below shows how the cross section of a coal entry and a nonmetal entry can differ and affect ventilation parameters:


**(https://canvas.instructure.com/courses/1190544/files/69793787/download?wrap=1)**

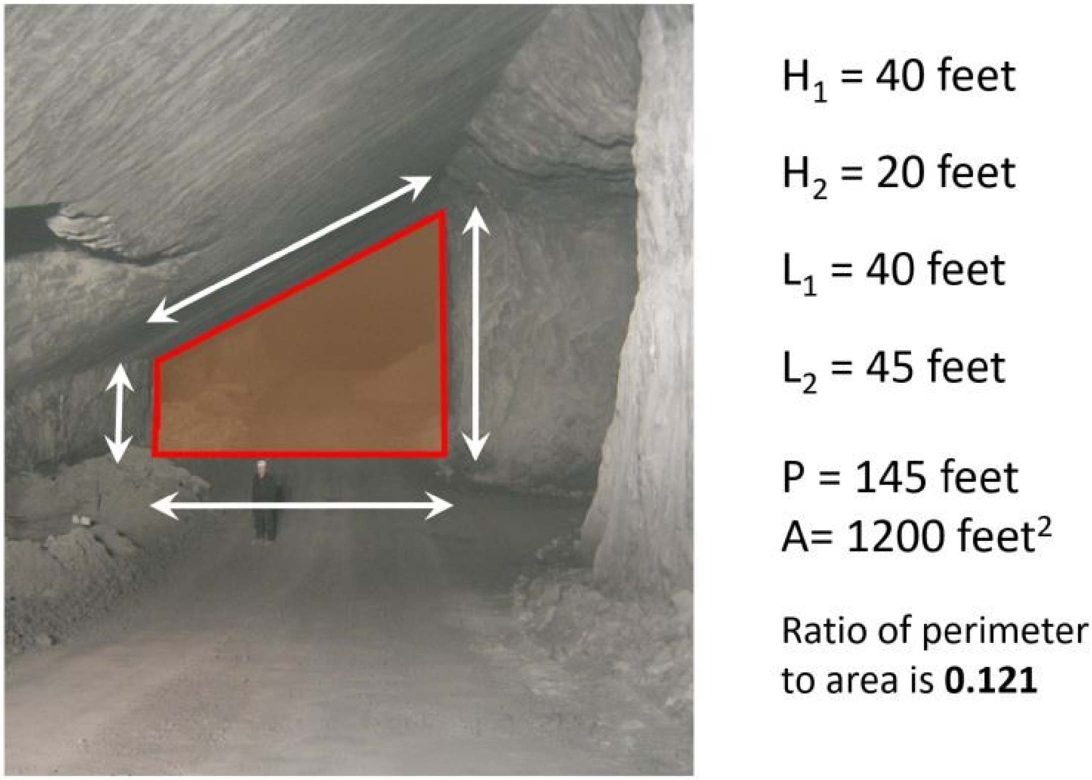



Figure 2.1. Parameters for coal mines and metal/nonmetal mines, effects on resistance.

Of course, there is no hard and fast rule for entry size, but coal mine entries do tend to have smaller cross sectional areas (often limited to a span of 18-22 feet), and higher velocities.

# 3.0 Coal Mining Methods

In the US, coal is exclusively mined by cutting. There are no mines blasting and loading coal underground. Cutting is accomplished via continuous miner, shearer longwall or plow longwall. Continuous miner machines, along with room and pillar methods have a variety of permutations, including:

**Longwall development** (no pillar recover, emphasis on careful geometry for later longwall mining)

**Room-and-pillar production** (no pillar recovery, similar in most ways to lonwall development except there may be more entries.)

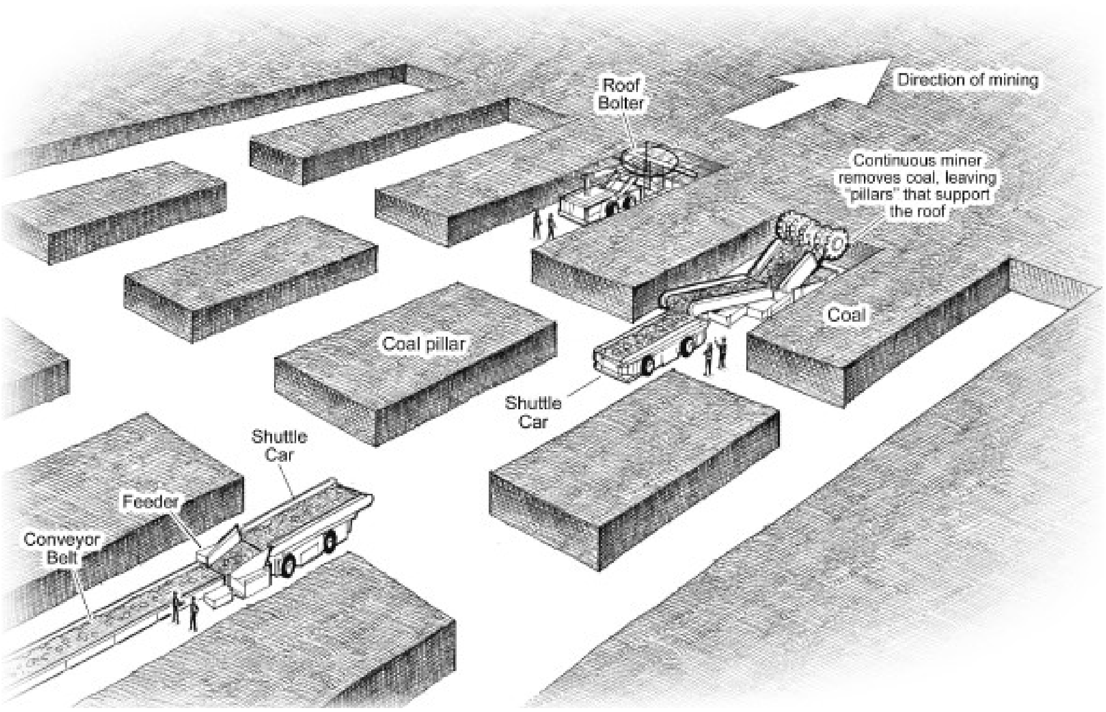



**Figure 3.1 Room and pillar development or production**

Room-and-pillar retreat (includes pillar recovery and ventilation of a gob)

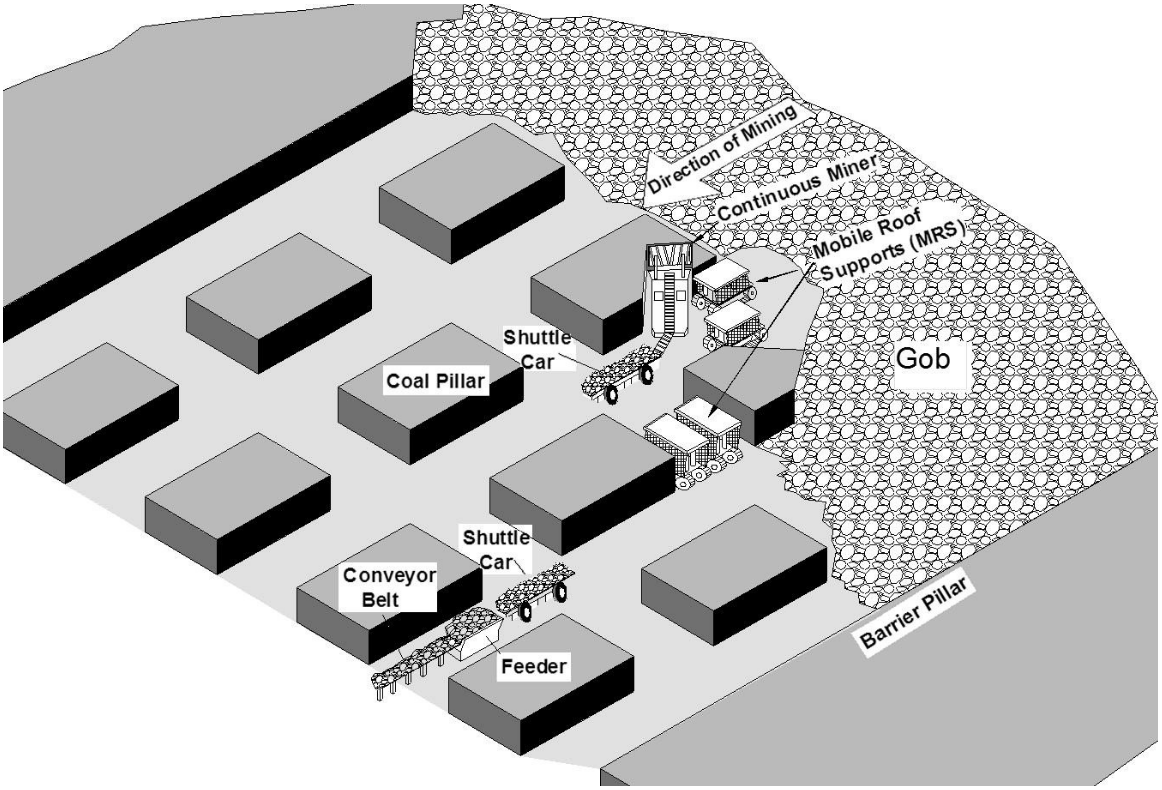



**Figure 3.2 Room and Pillar Retreat Mining Overview ([Mark, C. and Guana, M., 2017 (https://www.sciencedirect.com/science/article/pii/S209526861630204X)](https://www.sciencedirect.com/science/article/pii/S209526861630204X))**

**Shearer longwall mining** (coal is cut from a several mile by approximately 1,000 foot block by a mounted cutting machine with two rotating heads.)

**Figure 3.3. Longwall mining, shearing cutting drum to the left**, *notice the operator is wearing an air stream helmet"for protection against respirable dust. (NASA, 2015 (https://landsat.gsfc.nasa.gov/assessing-longwall-mining-impacts-on-forests-above/) ).*

**Plow longwall mining** (similar to shearer, except that coal is cut by a passive "plow" that has cutting bits and is pulled along the 1,000 foot face).

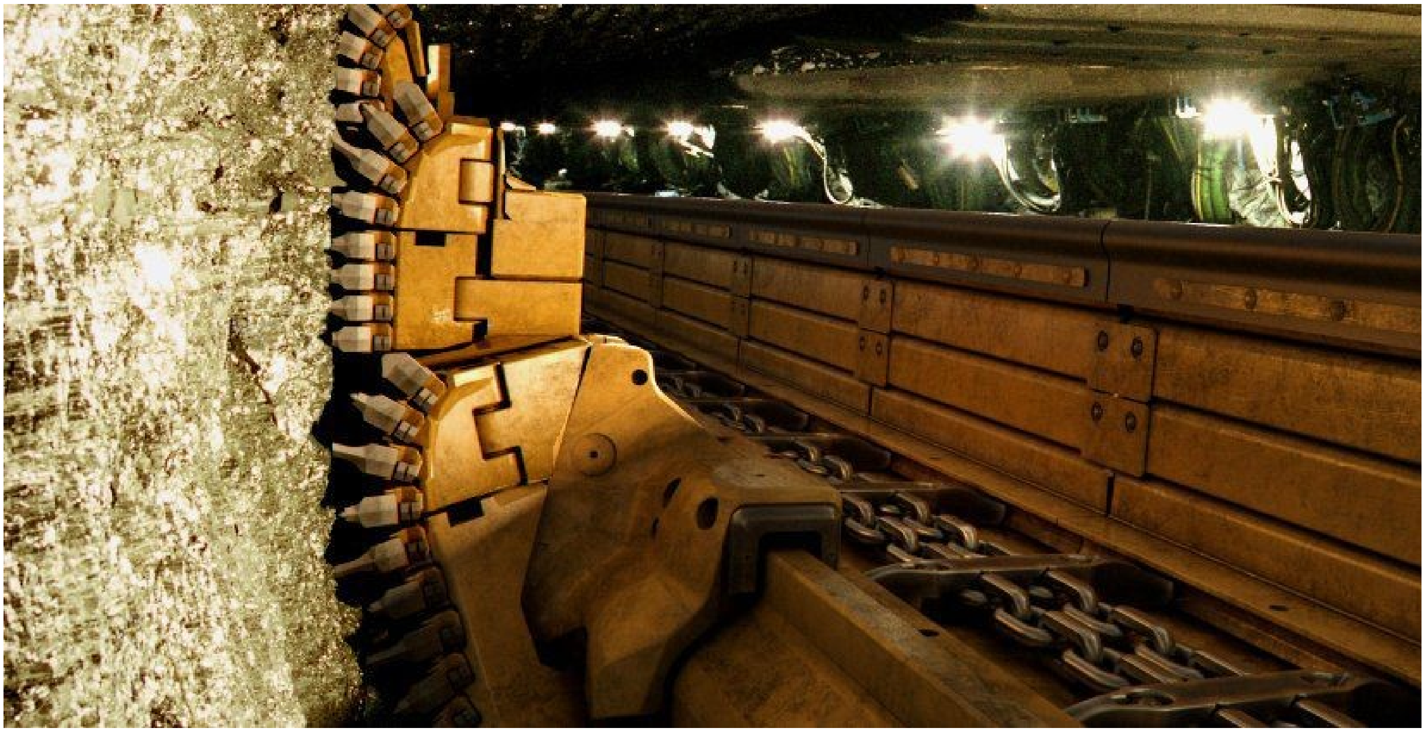


**Figure 3.4. Plow longwall.** The plow is shown to the left against the coal face, while the armoured face conveyor is shown towards the right. (**Caterpillar, 2018** **(https://www.cat.com/en_US/campaigns/awareness/underground-plows-low-seam-coal.html)** )

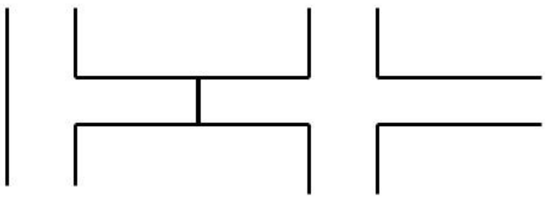
Learn more about longwall mining:


Aside from the mining method, other factors that influence ventilation design are over- and under- mining, whether previous or active. These works can cause communication between one mine or another allowing for inflow of water or gas. Precise and accurate surveying and monitoring are critical.

# 3.1 Ventilation Controls and Mapping

Below is a description of the most common coal mine ventilation controls along with common map symbols (may vary by state or company) and pictures when available.

**Temporary Stoppings (also referred to as temporary brattices)**



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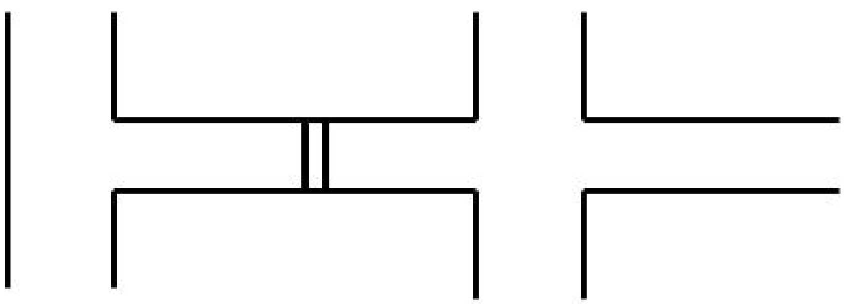
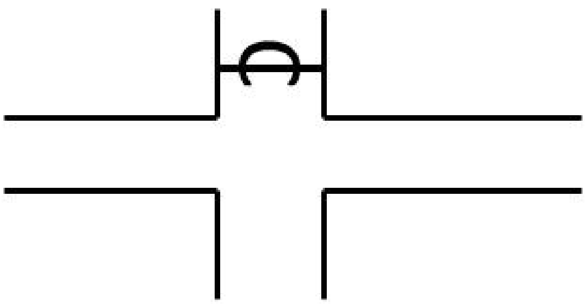
A temporary stopping is a temporary wall built across a mine passage to prevent airflow. They are usually made of a fire resistant plastic material, and may be supported by jacks or 2"x4" timbers wedged into an entry. The map symbol is usually a single line. They are erected until a permanent stopping can be built, and used in active workings where airflow may change. Figure 1 shows the map symbol for a temporary stopping, and a temporary stopping installed in a cross cut.

**Permanent Stoppings**



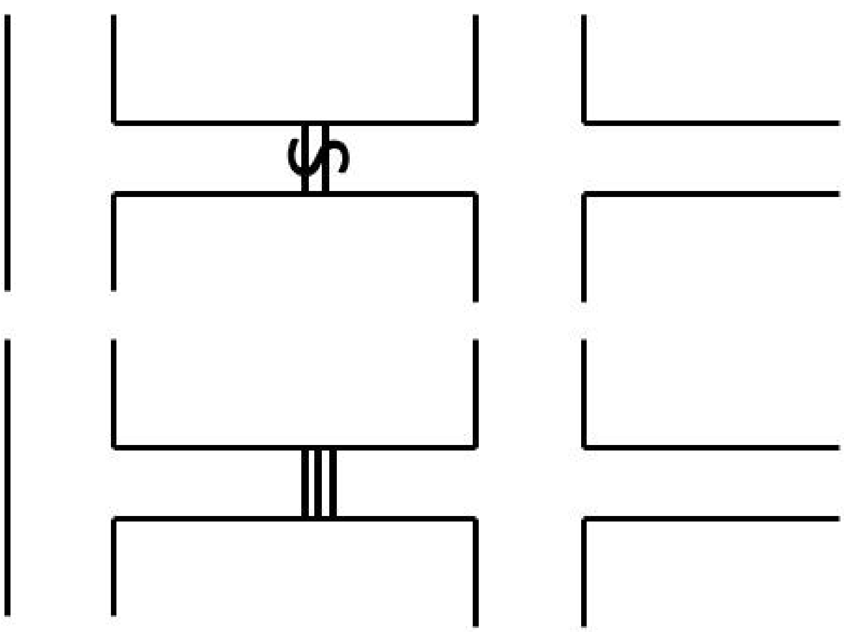
Permanent stoppings are intended to last for months or years. They may be be built from blocks or there are other engineering solutions, such as Kennedy stoppings. Block stoppings may be "drystacked" (no mortar between the joints), or they may be mortared between joints. Additionally, they are usually mortared on the high pressure side to help prevent leakage and wedged into the top. Depending upon geologic conditions stoppings can also be "keyed" in, meaning that they are installed in trenches in the floor or the floor, ribs and roofs, to assist with stability. In some strata the stopping may be airtight but communication pathways might develop in the roof, rib or floor that allow air leakage. Finally, ground conditions can significantly affect stopping maintenance, as stoppings may be crushed out or crack due to ground conditions. Man doors are routinely installed in stoppings so that miners can easily move from one entry to another and these may also be a source of leakage.

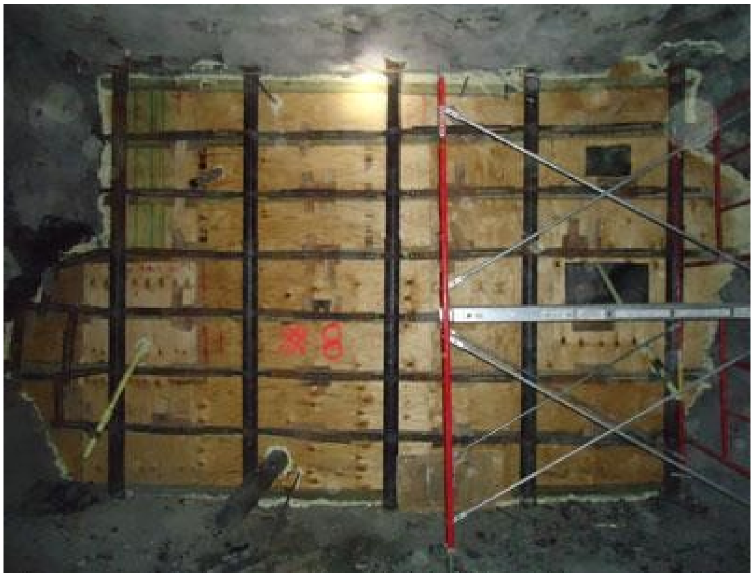
**Check curtains**



Check curtains are routinely used on active sections to control and direct air. They are typically only fixed to the roof to allow equipment to run through. In mines with high airflow, strips of conveyor belt are sometimes used over the check curtain to hold it vertical. These might also be referred to as "flypads". Leading practice dictates that clear check curtain be used on active sections and that equipment emit a clear auditory warning before running through a check curtain as these have been a common location for injury of a pedestrian by moving equipment.

**Seals**



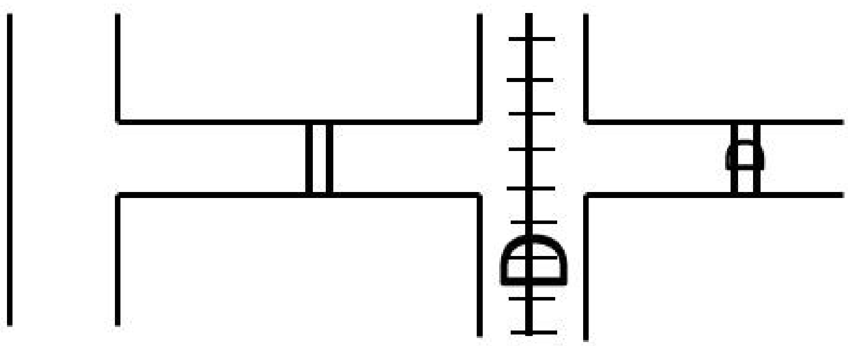


Seals are special permanent stoppings used to isolate abandoned workings. Design and construction of seals is heavily regulated in US coal mines. By sealing inactive areas, the ventilation engineer can improve safety, efficiency, and cut cost. Actively ventilated areas of underground coal mines in the US must be physically inspected once per week, and, because they are older areas, often have more associated hazards (loose top, rubble on the floor, etc), and higher leakage. Learn more about seals.

**Box checks or box stoppings**

These are stoppings (temporary or permanent) with a hole in them to allow a convey belt to run through.

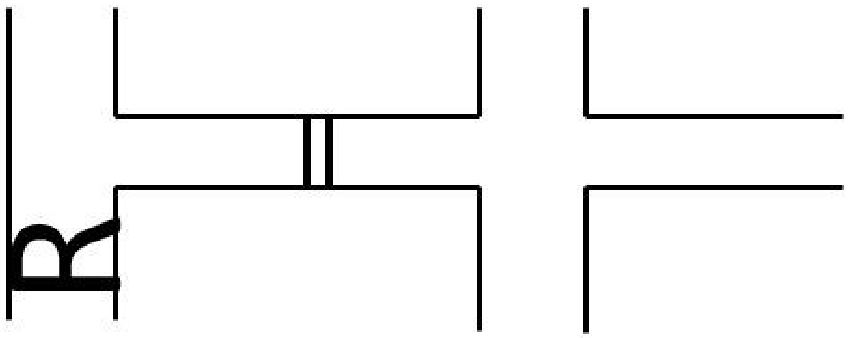
**Doors**



[(https://canvas.instructure.com/courses/1190544/files/54059796/download?wrap=1)](https://canvas.instructure.com/courses/1190544/files/54059796/download?wrap=1)

Doors can be manually or automatically operated. They may allow just a person through or cover an entire entry and allow equipment to run through. When large doors are installed they are often installed in a sequence of two, creating an airlock. This practice avoids short circuiting and also allows the doors to be opened in high pressure conditions. Man doors may also be installed in a airlock configuration, and this is most often done in areas with a high pressure differential where physically opening the door can be very difficult. These areas create a risk for injury as miner can be pulled through doors if the pressure differential is quite high.

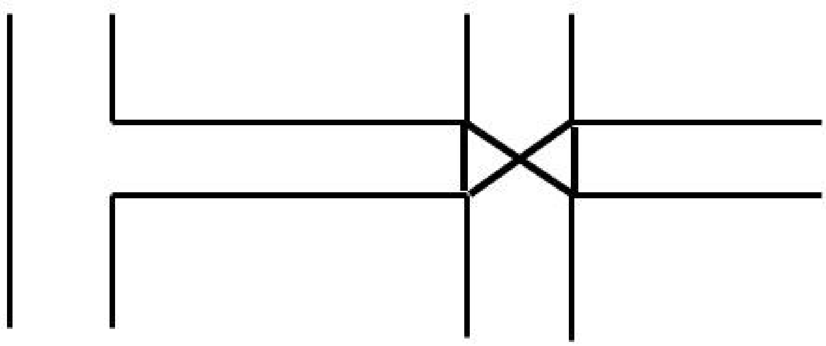
**Regulator**

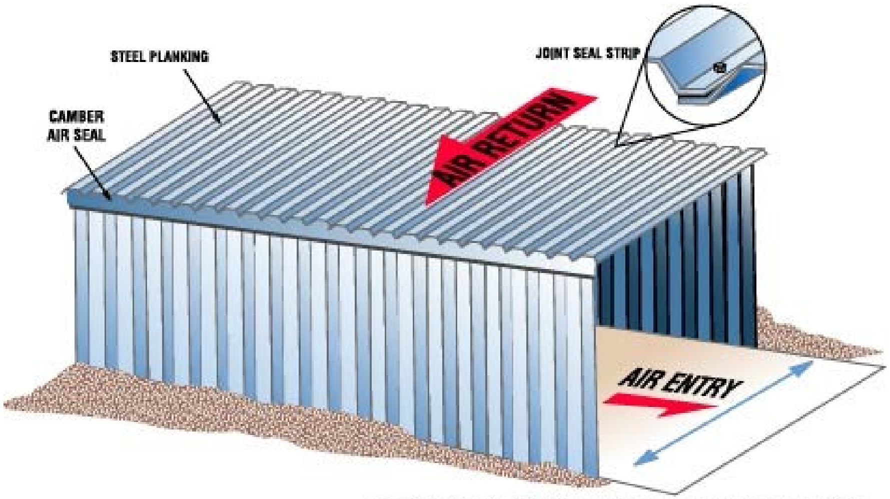



Regulators create a shock loss to reduce the passage of air and allow the the controlled flow of air from a mains to multiple sections. Regulators are like valves on a system of pipes. There are multiple types of regulators but the most common in underground coal mines are a set of adjustable, sliding partitions that can be varied to the desired opening size. They are usually located at the mouth of a section on the return side to minimize interference with traffic.

[Learn how to size a regulator. (https://canvas.instructure.com/courses/1190544/pages/how-to-size-a-regulator)](https://canvas.instructure.com/courses/1190544/pages/how-to-size-a-regulator)

**Overcasts**






INSIDE OVERCAST WIDTH IS ONLY 5" LESS THAN OUTSIDE

Overcasts are enclosed airways at an intersection which allow the intake air and return air to cross without mixing. When designing a ventilation system, engineers should aim to minimize the number of overcasts because they are costly to install and maintain, particularly in lower seams. They require that at least double the mining height be cut in a crosscut, which can be quite costly.

**Undercasts**

Undercasts serve the same function as overcasts, but they are cut into the floor. They are less common because there are often problems with rock, coal, and water accumulation.

**Fan and tubing**

Auxiliary fans (link) and tubing are often used to direct air into dead end entries. Additionally, curtain can be used to line an entire entry and pull air from the face to return. Tubing and fans may be slightly more costly to install and maintain, but often result in more efficient air in the face, particularly if the coal block being cut is long (>100 feet).

# How to size a Regulator

A regulator adds resistance (by shock loss) to an airway to control how much air flows down the airway.

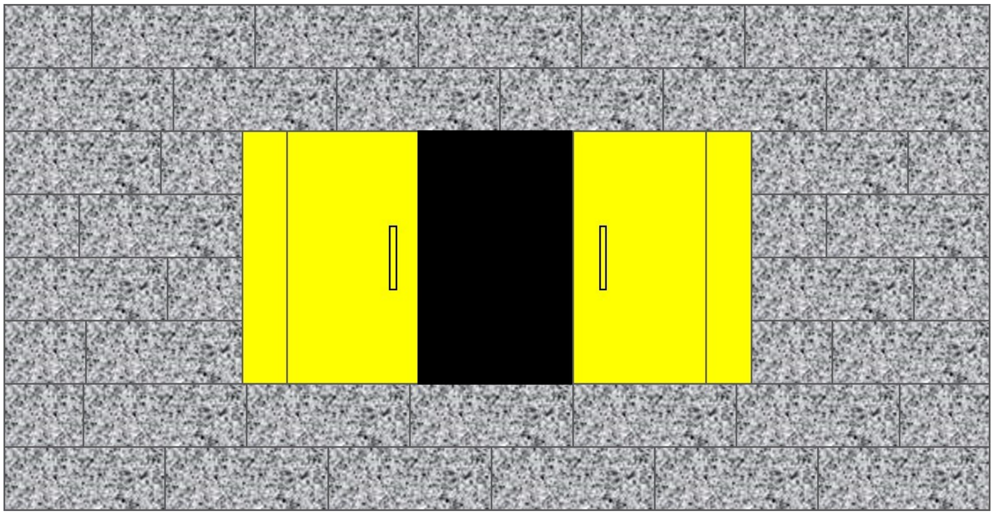


Figure X. Schematic of a square edge regulator with sliding doors, set in a block wall.

First, Calculate velocity head, $H_{v:}$

$$V = \frac{Q}{A} = \frac{225{,}000}{250} = 900 fpm$$

$$Mine\ H_v = \left(\frac{900}{4009}\right)^2 = 0.0504\ in.\ H_2 0$$

Next calculate shock loss (simulate your required shock loss):

$$X = \frac{H_x}{H_v} = \frac{1.0}{0.0504} = 19.84$$

Calculate N (for a square edge regulator, N = 2.5):

$$N = \sqrt{\frac{Z}{X + 2\sqrt{X} + Z}} = \sqrt{\frac{2.5}{19.84 + 2\sqrt{19.84} + 2.5}} = 0.283$$

[(https://canvas.instructure.com/courses/1190544/files/69795866/download?wrap=1)](https://canvas.instructure.com/courses/1190544/files/69795866/download?wrap=1)

Finally, N will give you an orifice area which indicates how far you should open the regulator doors.

$$N = \frac{Orifice}{Airway} = \frac{Orifice}{250} = 0.283 \rightarrow \boxed{71ft^2}$$

(https://canvas.instructure.com/courses/1190544/files/69795866/download?wrap=1)

# 3.2 Booster Fans versus Auxiliary Fans

A critical distinction in US underground coal mines is the difference between booster fans and auxiliary fans. Physically, a booster fan and an auxiliary may refer to the exact same piece of equipment, generally, a smaller more portable mine fan. However, in practice, booster and auxiliary fans serve entirely different purposes. Booster ventilation fans serve to boost the air supplied by a main mine fan. In other words, a booster fan is working in concert with a main mine fan to pull air into the mine. An auxiliary fan, on the other hand, is simply directing air that has already been moved into the mine by the main fan, generally into a dead end heading. This distinction is necessary because booster fans are not legal under US regulation (**30 CFR §75.302 (https://www.ecfr.gov/cgi-bin/text-idx?SID=1d3aaaefa8aa818ae77ce1a2759f8c24&mc=true&node=se30.1.75_1302&rgn=div8)**).

# 4.1 Face Ventilation (Continuous Miners)

The ventilation of coal faces via either forcing air into the dead end heading (blowing) or exhausting air from the dead end heading (pulling). This is most commonly accomplished via an auxiliary fan located outby the face and attached to tubing or ventilation curtain (brattice) sealed between the roof and floor approximately 1-2 feet from the rib.  Common ventilation schematics are shown below, along with advantages and disadvantages.

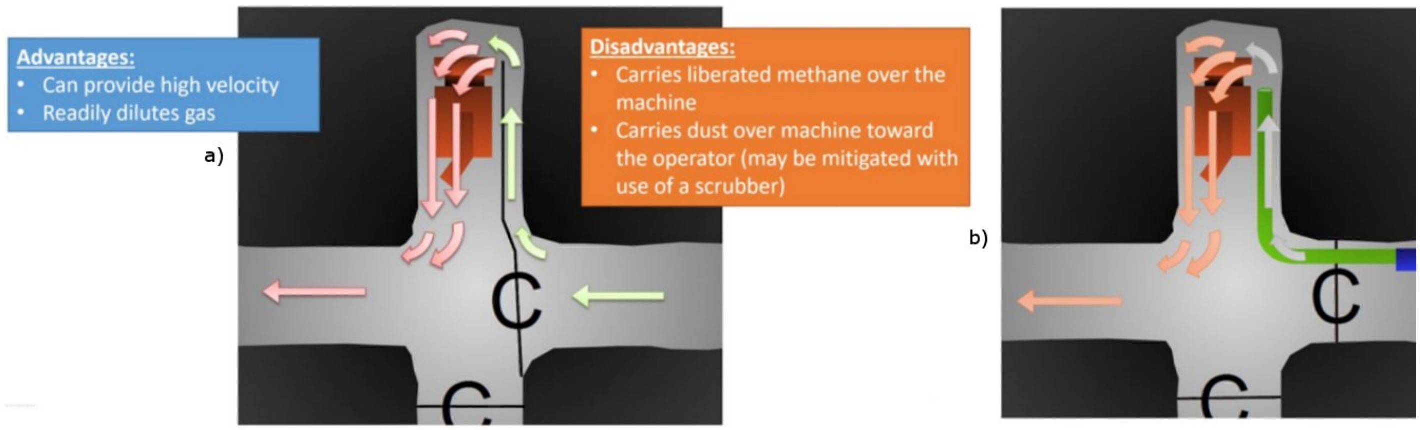



**Figure 4.1 Blowing face ventilation system (plan view).**

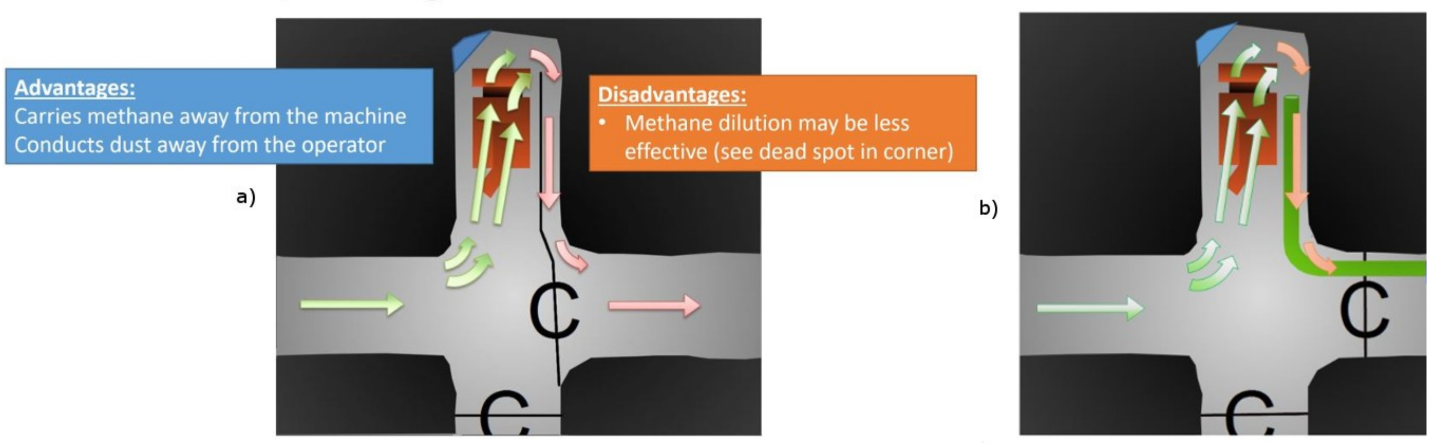



**Figure 4.2 Exhausting face ventilation system (plan view).**

# 4.2 Common Section Ventilation Design Plans

Presented below are several common ventilation schemes in active mining areas.

The first figures shows a unidirectional, or single split ventilation system on a development or room and pillar production section.

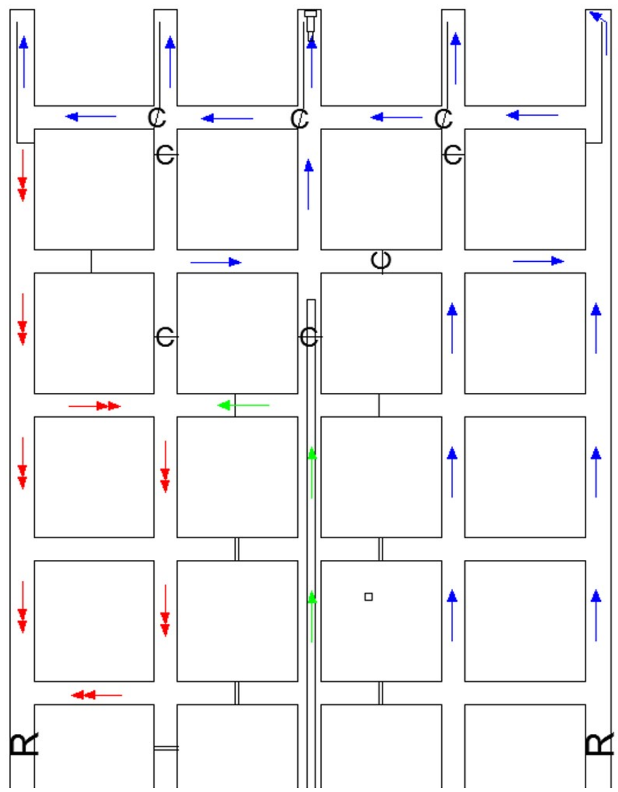
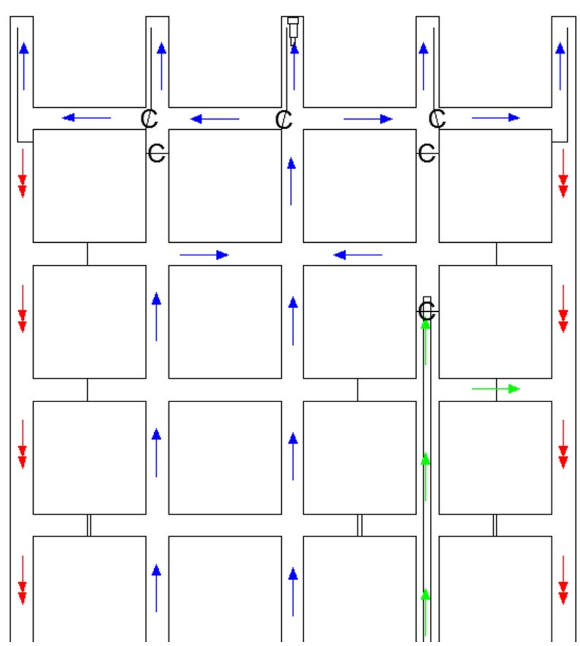


**Figure 4.3 Unidirectional or single split system.**

The single split system is best used in a mine with relatively little methane, as the intake air is flowing against a solid block of coal. In a particularly gassy mine that intake air may pick up considerable methane prior to reaching the face.

Next are two schemes for a dual split section (also referred to as bidirectional). Notice that a dual split will require the construction of more ventilation controls than a single split, but there are several reasons to choose this or a similar design.



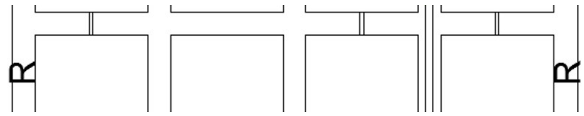



**Figure 4.4 A dual split or be directional system.** This system is more economical than the system in Figure 4.5 because it requires the construction of less stopping lines.

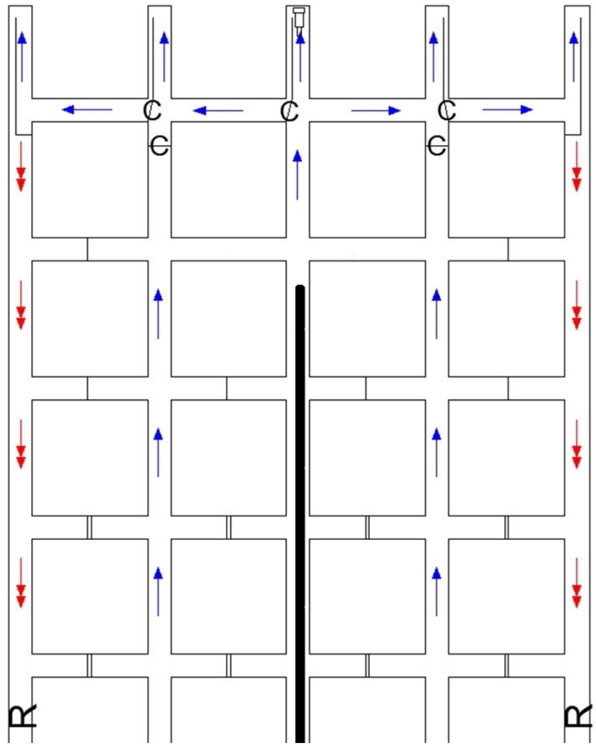



**Figure 4.5 A dual split or be directional system.**

Finally, a possible mouth of section ventilation schematic is shown.  The ventilation engineer must cross multiple air streams here while maintaining separation.  Ventilation to the section is also often controlled her via regulators.

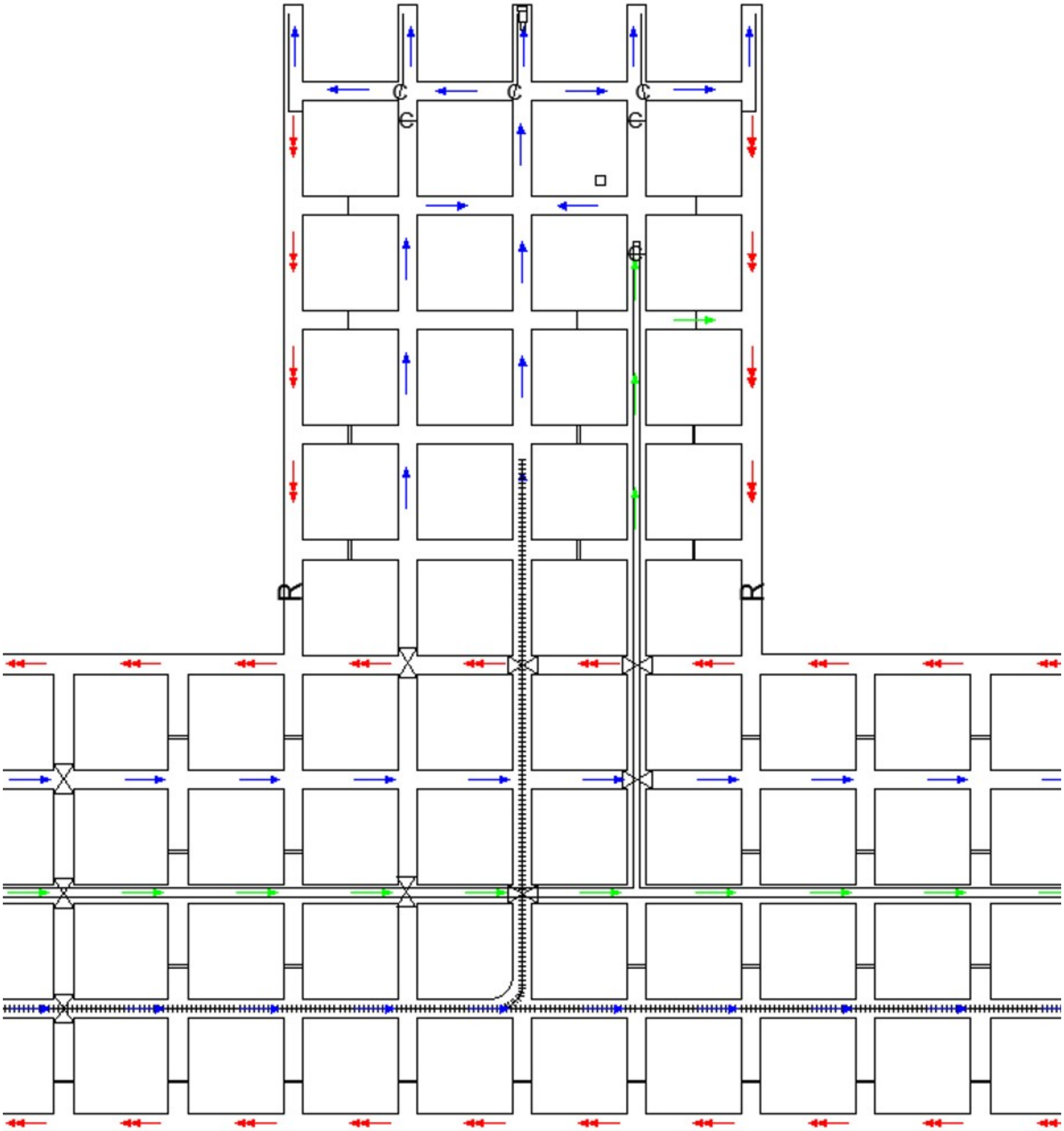



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**Figure 4.6. Possible mouth of section ventilation arrangment**

Below are two possible ventilation schematics for low methane and high methane longwall sections.

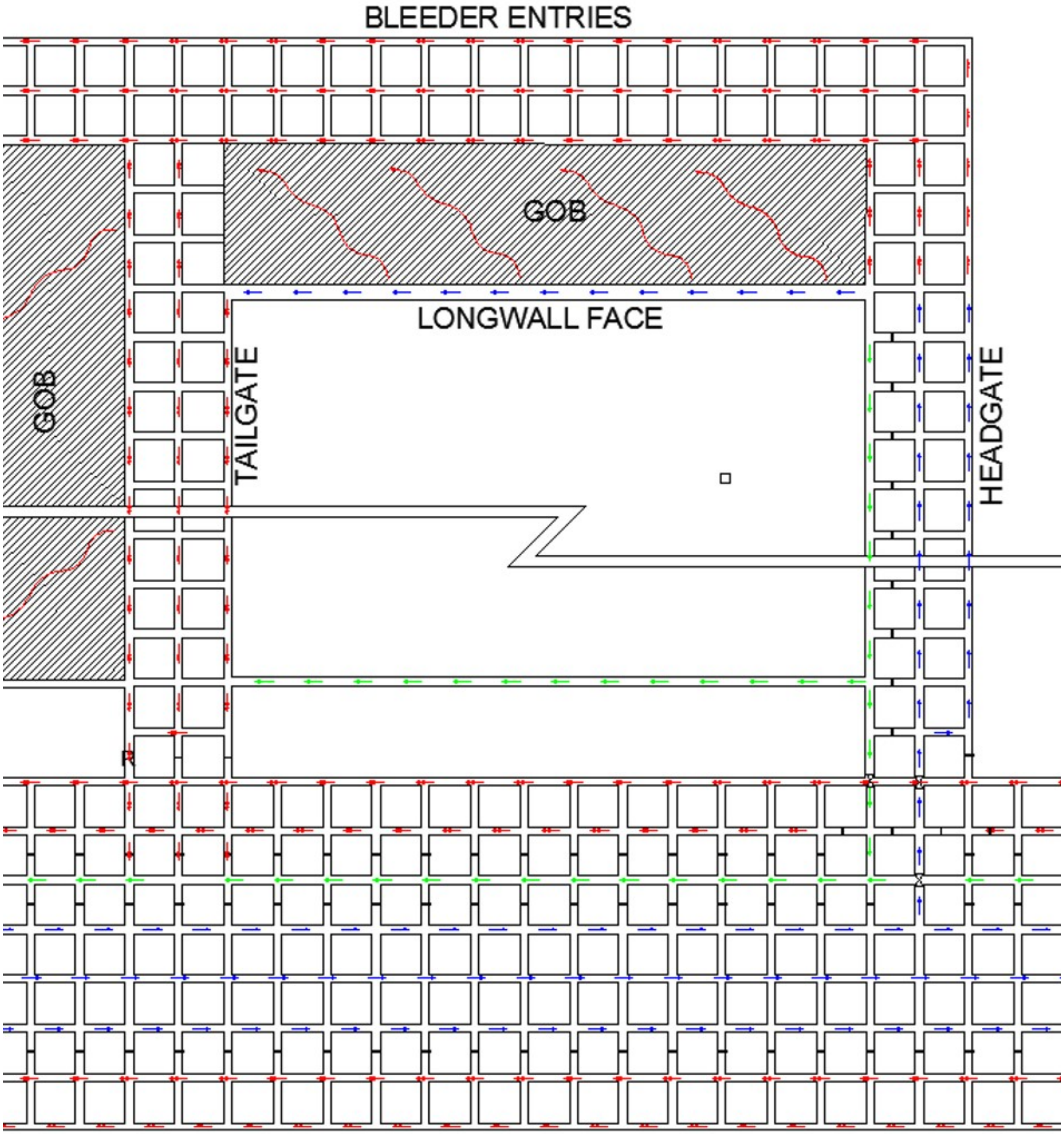


**Figure 4.7. An arrangement for a longwall section with low methane.** Notice that in this instance return air is moved down the tailgate.

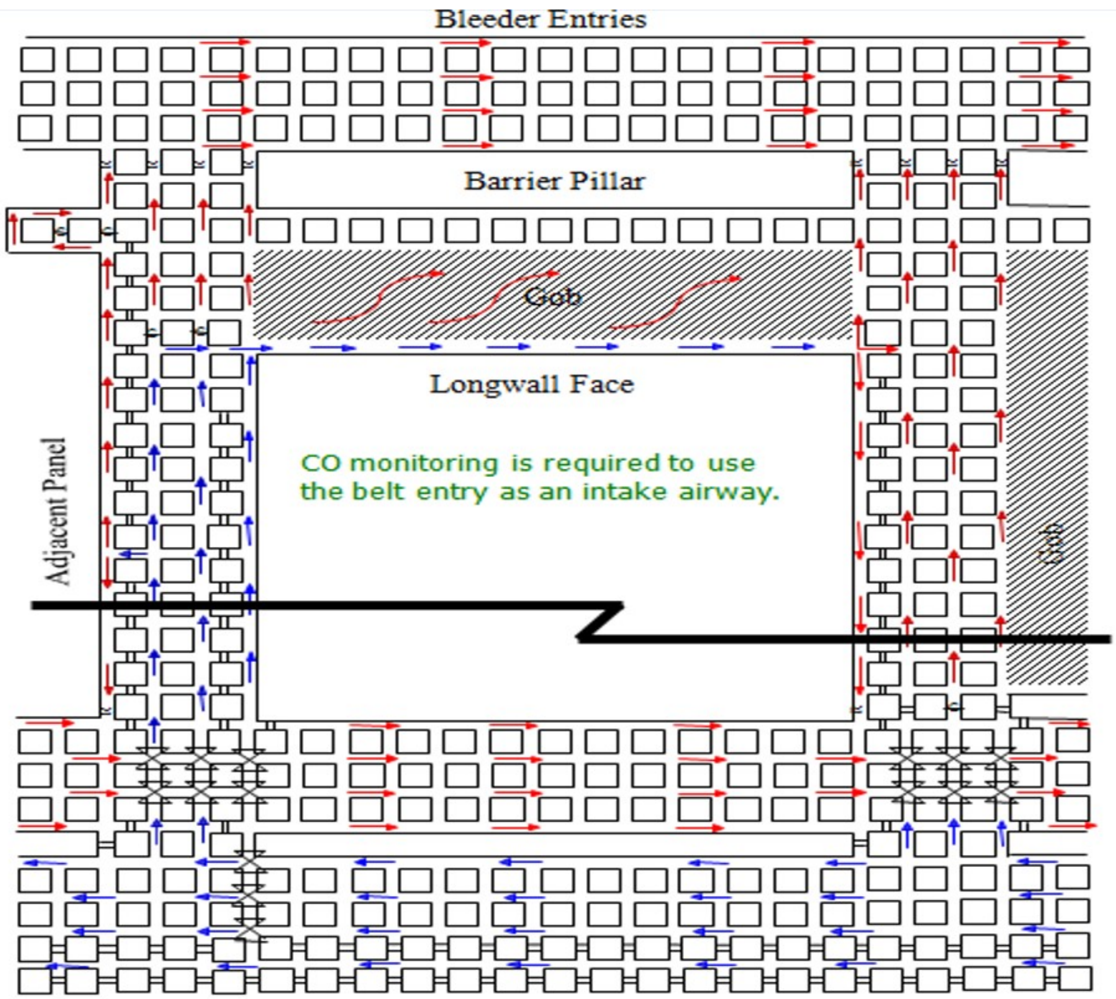


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**Figure 4.8. An arrangement for a longwall section with high methane.** Notice that in this instance additional air is supplied to the tailgate side, and the bleeder entries have a higher capacity.

Of course, every mine is unique with site specific design constraints, but understanding the conditions under which the schematics above might be utilized is a good starting point.

# Underground Metal/Non-metal Ventilation Design

[Jump to Today] [Edit]

**Metal/Nonmetal Mine Ventilation Design**

The potential variation in size and scope of metal / nonmetal mines around the world is almost infinite, and even individual mines will have very different demands for ventilation throughout the mine life. Mine location, geology, equipment and production rate are just some of the parameters that will influence the ventilation system design. In this course, we examine the parameters and processes that influence the design of ventilation systems for metal and nonmetal mines.

### Learning Objectives

1. **List parameters that influence ventilation system design.**
2. **Demonstrate how how to validate a ventilation model for use in ventilation system design.**
3. **Explain the difference between geologic, environmental and strategic design factors.**
4. **Identify various strategies for auxiliary ventilation.**
5. **Explain the mechanics of how airflow is used to dilute and remove contaminants.**
6. **Demonstrate how to select fans utilized in underground mine ventilation systems.**

Get Started

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# Course Summary:

| Date | Details |
|---|---|
| | MNM Ventilation System Design (https://canvas.instructure.com/courses/1049599/assignments/5491901) |

# 1.0 Basis of Design

Prior to beginning any ventilation design project, it is useful to ask “what is the point?” Establishing the purpose of your study can help you define both the start and end of the project, as well as what path you will take along the way. Defining the Basis of Design (BOD) is more than simply deciding on the design criteria (although that is an important part of the exercise). The reasons for conducting a ventilation design may vary greatly, from a simple “what if” exercise, conceptual design or detailed engineering study. The required inputs, and level of detail will also change proportionally. When executed correctly, a ventilation study can be an incredibly powerful predictive tool, that optimizes efficiency, and reduces risk, both physical and economic. However, an incorrect study, may actually hide risks, and give a false sense of security to those involved. Understanding the differences for various outcomes and the different levels of detail and inputs required is critical for the successful outcome of any ventilation planning project.

# 1.1 Introduction

As ventilation modeling tools (i.e. software packages) have evolved and become more powerful and user-friendly, a worrying consequence has also manifested itself in the practice of mine planning professionals. Although current iterations of popular mine planning software packages (e.g., Ventsim, VUMA and VnetPC) have all been proven reliable and useful planning tools, it should be remembered that their output is only as good as the inputs provided. Further to this point, it is also possible to misuse, or misinterpret even reliably accurate results if the user is ignorant, or inexperienced with regard to ventilation practice.

Dr. Rick Brake, founder of Mine Ventilation Australia has identified the following areas in which mistakes of this nature are most likely to occur:

- Failure of the principals to understand the project scope, and expected outcome(s) or deliverables.
- Failure of the study authors (designers) to use the appropriate inputs, assumptions and/or design criteria.
- Failure to develop or use a correct or *validated* ventilation model.

Dr. Brake goes on to surmise that although inexperienced ventilation practitioners are often guilty of making one (or more of these mistakes), these failures are not limited to just those ventilation engineers and technicians that are inexperienced or new to the subject matter. Even experienced, knowledgeable staff can make mistakes, if the impacts of various design assumptions or operating practices are not well considered or understood. In light of the significant negative impacts to health and safety and project economics that can occur when incorrect ventilation designs are implemented, it is critical for ventilation planners to adopt the following standards of practice with respect to ventilation system designs:

- Always use a properly correlated, or validated ventilation model as the basis for performing and predictive design.
- Always prepare an appropriate BOD report prior to beginning a ventilation project or study.

It is important to note that a properly validate model does not necessarily indicate a properly ventilated mine, or even a properly designed one. In fact, this is simply an indication of how accurately the ventilation model predicts the behavior of the actual ventilation system.

More detailed information about constructing and validating ventilation models may be found in the "Ventilation Modeling and Network Simulation" course attached to this series.

It is unfortunately not uncommon for a well-correlated ventilation model to accurately predict the behavior of a very poor ventilation system and/or design. Thus, in order to ensure the successful completion of any modeling exercise, the BOD must be effectively executed.

Ultimately, the success of the study or project will depend upon the following critical criteria:

1. A well-defined and understood scope of work including all desired outcomes and deliverables.
2. A properly validated ventilation model.

3. A properly constructed and documented BOD.
4. Execution by competent, experienced ventilation professional(s).

When the above criteria are all satisfied, then the probability of successful outcome for the ventilation study has been maximized.

Click on the link below to download Dr. Brake's paper.

**QA_Ventilaition_Model_Design_Brake.pdf (https://canvas.instructure.com/courses/1049599/files/46434948/download?wrap=1) (https://canvas.instructure.com/courses/1049599/files/46434948/download?wrap=1)**

# 2.0 Environmental Design Factors

Underground mines may be found in all parts of the world, and with conditions equally varied. From temperate rainforests to dry deserts, and from frozen arctic tundras to equatorial forests with all conditions in between. It is the responsibility of the ventilation engineer to both recognize how these varied environmental conditions will affect the mine ventilation system and formulate a design that will appropriately mitigate any hazards encountered. In this section we will explore some of these environmental factors, and how they impact the design of a mine's ventilation system.

# 2.1 Arctic (Sub-freezing) Environments

In arctic climates (or in any sub-freezing environment), the mine workforce may be exposed to potentially dangerous combinations of air temperature and velocity.  In some cases, temperatures and/or air velocities that seem innocuous by themselves may become deadly in combination.

Mine workers in sub-freezing environments may be at risk of *hypothermia* (an extreme drop in the body's core temperature) or *frostbite* (freezing of the flesh) if appropriate measures to protect them are not taken.

Elevated air velocities in sub-freezing temperatures increase the cooling rate of exposed personnel. *Equivalent Wind Chill Temperature* (Wind Chill) is defined as the ambient temperature in an airstream moving at 1.8 m/s giving the same rate of cooling as the actual temperature and air velocity.

Figure 2.1 shows the Wind Chill temperature at various ambient temperatures and airflow velocities.

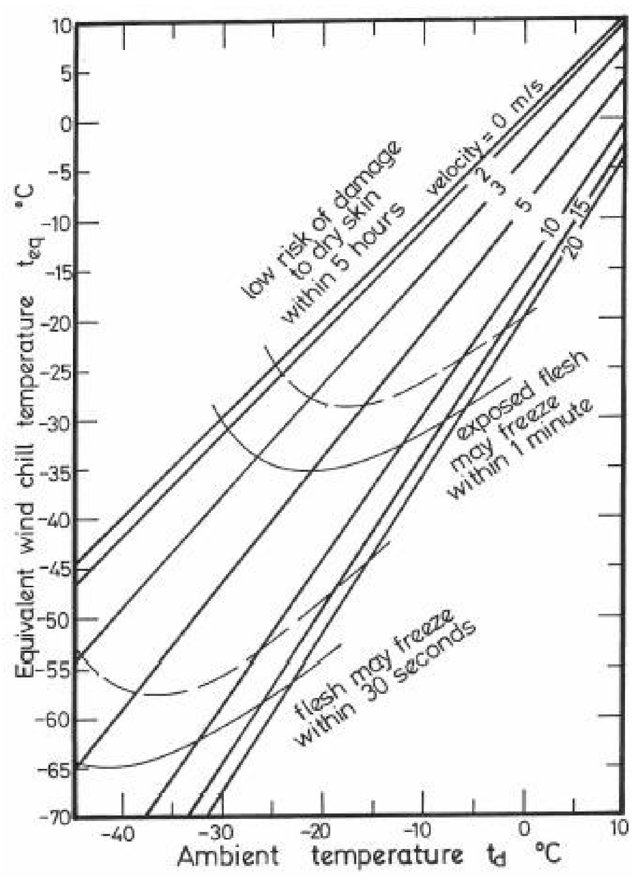



**Figure 2.1: Equivalent Wind Chill Temperature – US Army Research Institute for Environmental Medicine.**

Figure 2.2 depicts the relative danger of freezing flesh (frostbite) for various temperature/air velocity combinations provided by the American Conference of Governmental Industrial Hygienists (ACGIH).

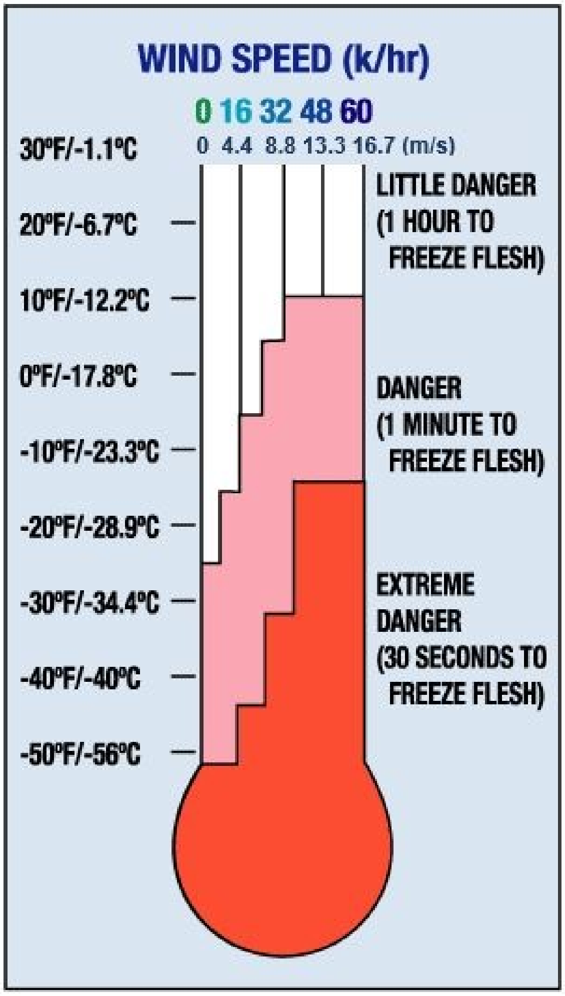



**Figure 2.2: Frostbite Danger (ACGIH TLVs and BEIs).**

In climates where the ambient air temperature can fall below 0 °C, air heating may be required.

In order to protect the integrity of the ground support system, constant freezing and thawing of the water within the rock strata should be prevented with bulk air heaters as necessary.

Under such conditions, ice may form and build up at un-heated mine portals, which can provide a hazard to both personnel and equipment traveling through them.

Cold temperatures have also been associated with reduced mental capacity/poor judgement and reductions in muscle coordination and productivity. Any mineworkers who will be exposed to sub-freezing conditions should be informed of the associated health risks and properly trained in appropriate techniques for avoiding injury.

Acclimatization to cold temperatures is possible, and should be implemented for new workers, but should not be relied on by itself to reduce the health risks associated with working in cold-weather.

At a minimum, miners working in sub-freezing environments should be trained to:

- Dress properly for conditions and in layers
- Work in pairs
- Take warming breaks as necessary

- Eat an appropriate cold-weather diet (calorie appropriate)
- Recognize the signs of cold-stress
- Perform first aid and treat any cold-related injuries that could be encountered

If bulk air heating is required, a study should be conducted to identify the appropriate means from the many available options for heating mine intake air; e.g., direct heating, indirect heating, waste heat recovery, latent heat addition (ice production) and geothermal heating.  Often, the best solution may include some combination of these methods optimized for reliability and cost-effectiveness.

More detailed information about the design and implementation of bulk air heating systems for use in mining applications may be found in the “Bulk Air Heating and Cooling” course attached to this series.

# 2.2 Tropical (Hot/Humid) Environments

Mines that are located in hot and/or humid climates pose very different, albeit equally serious risks to mine workers' health as those that are located in frigid climates. Heat and humidity in combination, and especially in the absence of adequate airflow velocity can cause various physiological problems for those exposed, up to and including, death.

Identifying and controlling potentially hazardous climatic conditions prior to their affecting the workforce is one of the principal responsibilities of the ventilation engineer.

*Heat Stress* is the affect of the environment upon the body, whereas *Heat Strain* is the body's reaction to the stress exerted on it by that environment. In a simple example, the elevated temperature and humidity inside an active development heading when the auxiliary fan is not turned on is an example of Heat Stress. The elevated core body temperature, profuse sweating and increased heartrate of the mine worker in this heading are an example of Heat Strain.

Heat illnesses experienced by underground miners can vary widely, from mild discomfort, cramping and both mental and physical fatigue in mild cases, to extreme cases such as renal failure, heat stroke and death.

If acclimatized, given adequate water, clothing and sufficient rest, healthy workers are able tolerate any "naturally occurring" climatic conditions; however, underground mines can often produce environments that will exceed the human body's tolerance for heat stress. This is a result of various factors that may include heavy, long-sleeved clothing, PPE such as respirators, helmets and boots, the limited availability of cool drinking water, and high levels of physical exertion that are often required for relatively extended periods of time.

Mine workers that may be exposed to elevated combinations of temperature and humidity underground should be trained to identify symptoms of heat stress in themselves and others. The mine should provide adequate water and ice for cooling where necessary, and develop an appropriate schedule governing work and rest periods or cycles.

Appropriate criteria for heat stress management should be part of the basis of design for any mine where the risk for heat-related illnesses has been identified. These criteria should include a Wet-bulb or Wet-bulb Globe temperature scale that governs the ratio of work/rest cycles as well as a maximum temperature at which all work is stopped and personnel are withdrawn from the area. Since this temperature is highly dependent on acclimatization and cultural norms, there is no universal standard for temperature criteria- nonetheless, this is an important design consideration that should be standardized within a particular mine, or mining company.

In some mines, it may not be possible to meet the required thermal criteria solely with ambient air, particularly in hot and/or humid environments. Many underground mines have surface conditions that come close to, or exceed the thermal criteria established as the upper limit for safe working conditions. In these cases, refrigeration of the mine environment may be necessary.

The application of mine air cooling can vary greatly in scope, application and cost, from air conditioned vehicle cabins to spot coolers designed to cool a specific heading to bulk air refrigeration plants that produce tens of Megawatts of cooling through chillers, solid ice or slurry.

Large-scale refrigeration plants are complex and expensive, and in some case the cost of bulk air cooling exceeds that of the entire ventilation system (minus the refrigeration). For this reason, bulk air refrigeration plants should always be designed by engineers with qualifications and experience in this field.

Specialized software packages designed to assist mine ventilation engineers and planners in predicting the ambient conditions underground should be used to ensure that the design criteria for the project are met and that the complex interaction of heat sources and installed cooling is properly understood.

Additional information about the design and implementation of bulk air refrigeration and cooling systems may be found in the “Bulk Air Heating and Cooling” course that is part of this series.

# 2.3 Alpine (High-Altitude) Environments

In mines that occur above 2000 m, there are several impacts to the ventilation system that need to be accounted for by the ventilation engineers and technicians responsible for designing, implementing and maintaining those systems. These impacts include the alteration of fan performance (relative to the manufacturer's standard operating curves), a change in diesel engine performance and emissions profiles, greater temperature fluctuations (diurnal and seasonal) and reduced oxygen content of the ambient air.

The theoretical pressure developed by a fan is derived from the rotational and tangential velocities of the impeller. Mathematically, this is expressed through *Euler's Equation*:

Euler's Equation:

$$p_{ft} = \rho u_2 C_{u2}$$

where: $p_{ft}$ = fan total pressure (Pa)

$\rho$ = fan air density (kg/m$^3$)

$u_2$ = peripheral speed of the blade tip (m/s)

$C_{u2}$ = tangential fluid velocity (m/s)

Euler's Equation also clearly demonstrates that the pressure developed by the fan is directly proportional to air density at the inlet. This relationship allows one to theoretically determine the operating curve for a given fan at any air density provided that the operating curve associated with any known air density is provided. Practically, this is accomplished through the use of the following equation:

$$\frac{p_{ft1}}{p_{ft2}} = \frac{\rho_1}{\rho_2}$$

where: $p_{fta}$ = fan total pressure at point 1 (Pa)

$p_{ftb}$ = fan total pressure at point 2 (Pa)

$\rho$ = fan air density at point 1 (kg/m$^3$)

$\rho$ = fan air density at point 2 (kg/m$^3$)

The "Fan Laws" are a general series of similarity laws that apply to all types of turbomachinery. They are presented in various formats in various sources, but always explain the relationship(s) between the performance variables of any two fans that have similar flow conditions.

Additional information regarding the Fan Laws and fan performance in general may be found in the Fans Course associated with this series of educational modules.

Most fan manufacturers provide characteristic operating curves for a given fan at standard air density of .075 lb/ft$^3$. In cases where fans will be operating at a density other than that for which the characteristic operating curve is known (i.e., at higher elevations) the *Fan Laws* can be utilized to predict the new operating points.

Practically, this can be used to approximate the performance of a fan at the actual inlet conditions where it will be operating provided that the “standard” density fan curve is known.

**Example:**

XYZ Mining company specifies a new fan for their mine, located at approximately 3000 m above sea level, where the air density is approximately 0.051 lb/ft$^3$. The response they receive from the fan manufacturer includes the following fan curve demonstrating the fan performance at standard density (0.075 lb/ft$^3$). Will this fan meet the required fan performance target?

Table 2.1 shows the conversion of operating pressures based upon the ratio of air densities (note that the airflow is held constant in this case).

**Table 2.1: Fan Curve Conversion for Altitude (Density) Changes**.

| Point No. | Standard Density* Pressure Drop (m in w.g.) | Quantity (kcfm) | Mine Density** Pressure Drop (m in w.g.) |
|---|---|---|---|
| 1 | 11.7 | 50 | 8.0 |
| 2 | 11.0 | 61.5 | 7.5 |
| 3 | 10.0 | 69 | 6.8 |
| 4 | 9.0 | 74.5 | 6.1 |
| 5 | 8.0 | 78.5 | 5.4 |
| 6 | 7.0 | 82.5 | 4.8 |
| 7 | 6.0 | 86 | 4.1 |
| 8 | 5.0 | 89 | 3.4 |
| 9 | 4.0 | 92 | 2.7 |
| 10 | 3.0 | 95 | 2.0 |
| 11 | 2.0 | 97.5 | 1.4 |

* 0.075 lb/cu. ft
** 0.051 lb/cu. ft

Figure 2.3 depicts these two fan operating curves at their respective air densities.

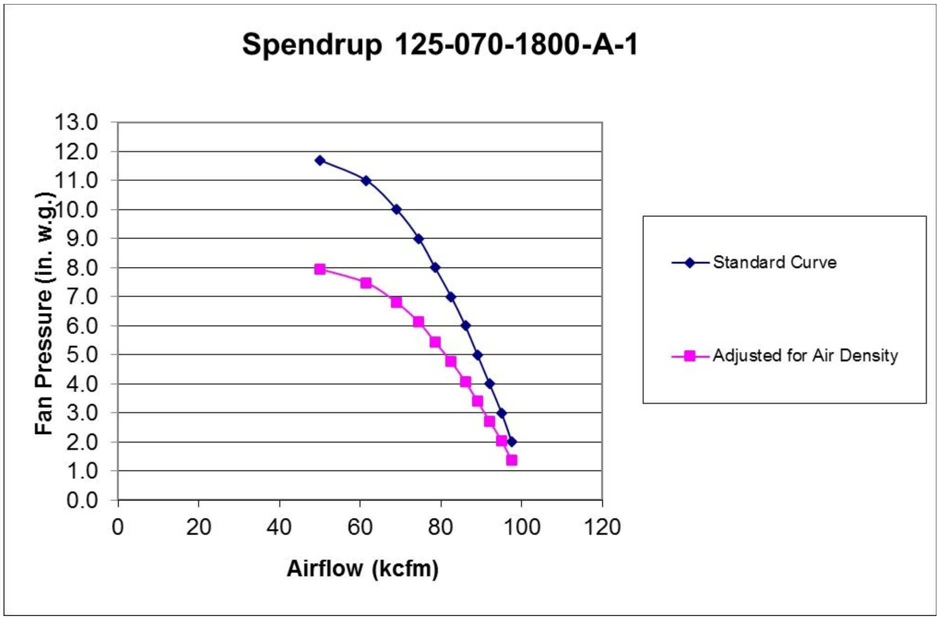



**Figure 2.3: Fan Curve Conversion for Altitude (Density) Changes.**

Whether designing the ventilation system for a new mine, or making changes to the ventilation of an existing mine, it is critical to align the air density utilized to develop the resistances of the network (and model) with the characteristic fan operating curve(s). Failure to do so may result in the selection, purchase and installation of a fan or fans that will not meet the projected (and required) fan duty.

Figure 2.4 shows once such example, with the projected operating point at standard density shown in red, and the actual operating point shown in yellow.

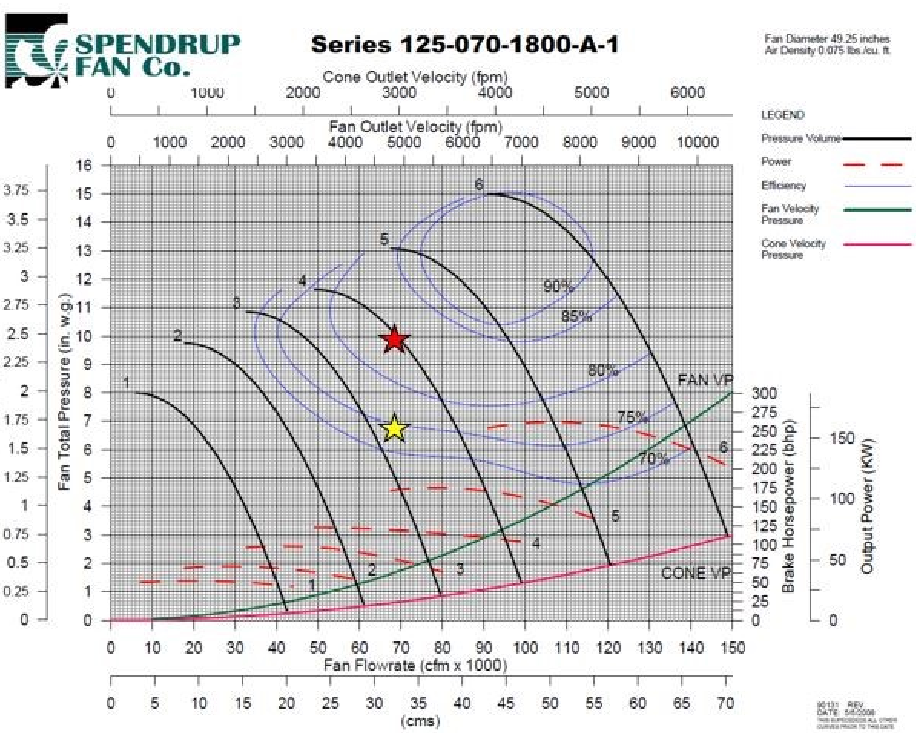



**Figure 2.4: Fan Operating Point Adjusted for Air Density Adjusted for Altitude.**

Another way in which altitude can affect the ventilation system is by altering the operation of the mine's diesel equipment fleet. Diesel equipment will often operate inefficiently, and may even be permanently damaged if it is not properly adjusted for the appropriate air density.

Most equipment manufacturers provide *engine derate curves* for properly making such adjustments. It may be necessary to correct the settings for each model or series of engines in use.

Figure 2.5 depicts a typical engine derate curve. In the absence of a manufacturer-provided and engine-specific curve, engine power should be derated by approximately 3% for every 300 m in elevation above sea level.

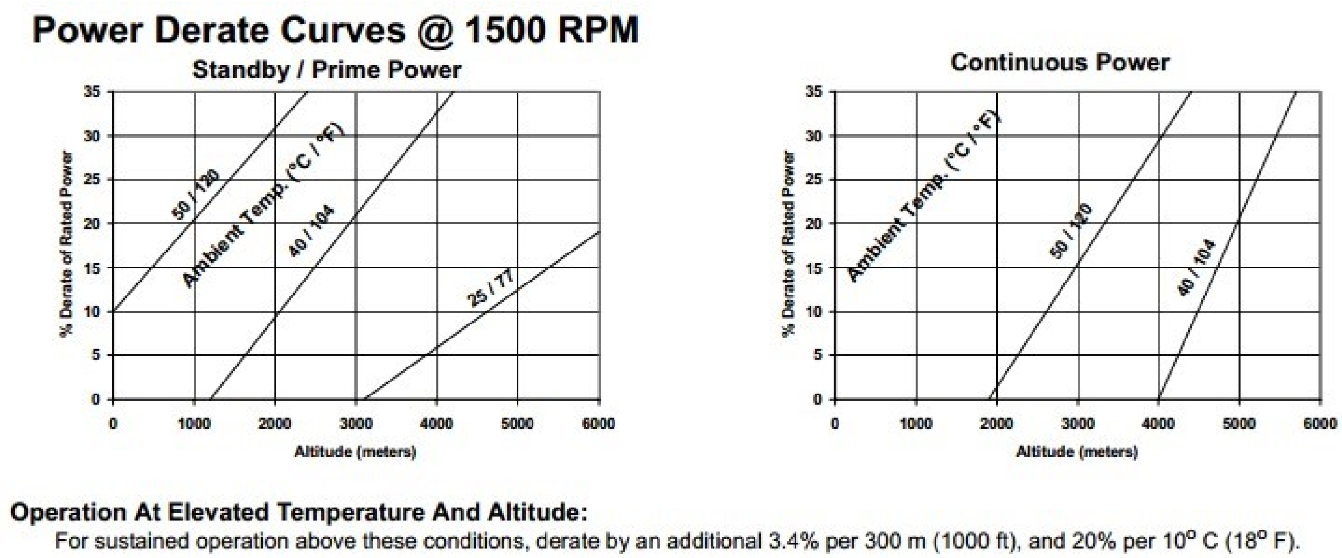



**Figure 2.5: Engine Derate Curves for Cummins QSK23-G3 NR1 (Cummins, 2003).**

As long as the impacts from high altitude are understood and incorporated into the design of the ventilation system, the ability of the system to meet the needs of the project should not be negatively impacted.

# 2.4 Regulatory Environment

Another critical impact on the mine ventilation system based on its physical (geographic) location is the determination of regulatory oversight for that mine and ventilation system.

The regulations that govern mines, and mine ventilation in particular will vary widely based on the country in which that mine is located, and in some cases, the state or province within that country.

These differences can in some cases be quite dramatic, with variations not only in what is allowable, but in the level of exposure of contaminants, and even in what ventilation strategies and/or hazard mitigation techniques are allowed.

For example, some regulatory bodies place limits on ambient levels of contaminant exposure based on highly specific PEL or TLV values, while other regulators may dictate a fixed airflow that is required in a certain area, or for a specific piece of equipment, while others still may simply limit the minimum airflow velocity of an area or drift.

Some examples of ventilation system regulations include, but are not limited to:

- Location of primary fans
- Frequency of environmental or equipment monitoring
- Storage and transport of flammable materials
- Workshop ventilation
- Contaminant exposure

A thorough understanding of all applicable laws and regulations is a required procedure when designing any ventilation system for new or existing mines.

# 2.5 Demographic Environment

Local demographics can also have an impact on the mine ventilation system. Variables such as the population density, proximity and familiarity with mines and mining culture can all affect how the mine ventilation system is perceived and tolerated, and may dictate how many and where mine fans are located.

Noise, in particular, is one factor that has proven to be problematic for many mining operations that operate in proximity to other industrial and non-industrial populations. In some cases, even isolated dwellings can cause disruption to the mine ventilation plan if their interactions are not properly addressed.

Noise from mine fans is generally unwelcome by the population at large, and in some cases, negative health impacts have been observed in those exposed to sources of low-frequency noise, even when that noise lies outside the range of what is perceptible to the unaided human ear. Nonetheless, this is an issue which must be addressed when the mine fans will be audible to persons outside of the mine property.

The negative health effects resulting from exposure to low frequency noise are not well understood, and conclusive information on the subject is not readily available (Findeis and Peters, 2004). However, there are many examples of mines that have encountered resistance from local populations that were disturbed by noise coming from the mine fans.

In cases where residential dwellings or other noise-sensitive locations (e.g., schools, wildlife sanctuaries, etc.) lie in close proximity to the mine, noise reduction strategies should be considered. These may include locating principal fan installations underground, installing silencers and other sound-deadening insulation and directing discharge evasés away from any nearby residents. Even the type of fan selected (i.e., axial vs. centrifugal) can have an impact on the noise produced by the installation.

Figure 2.6 shows the noise reduction achieved by various silencers available for a single axial-type fan.

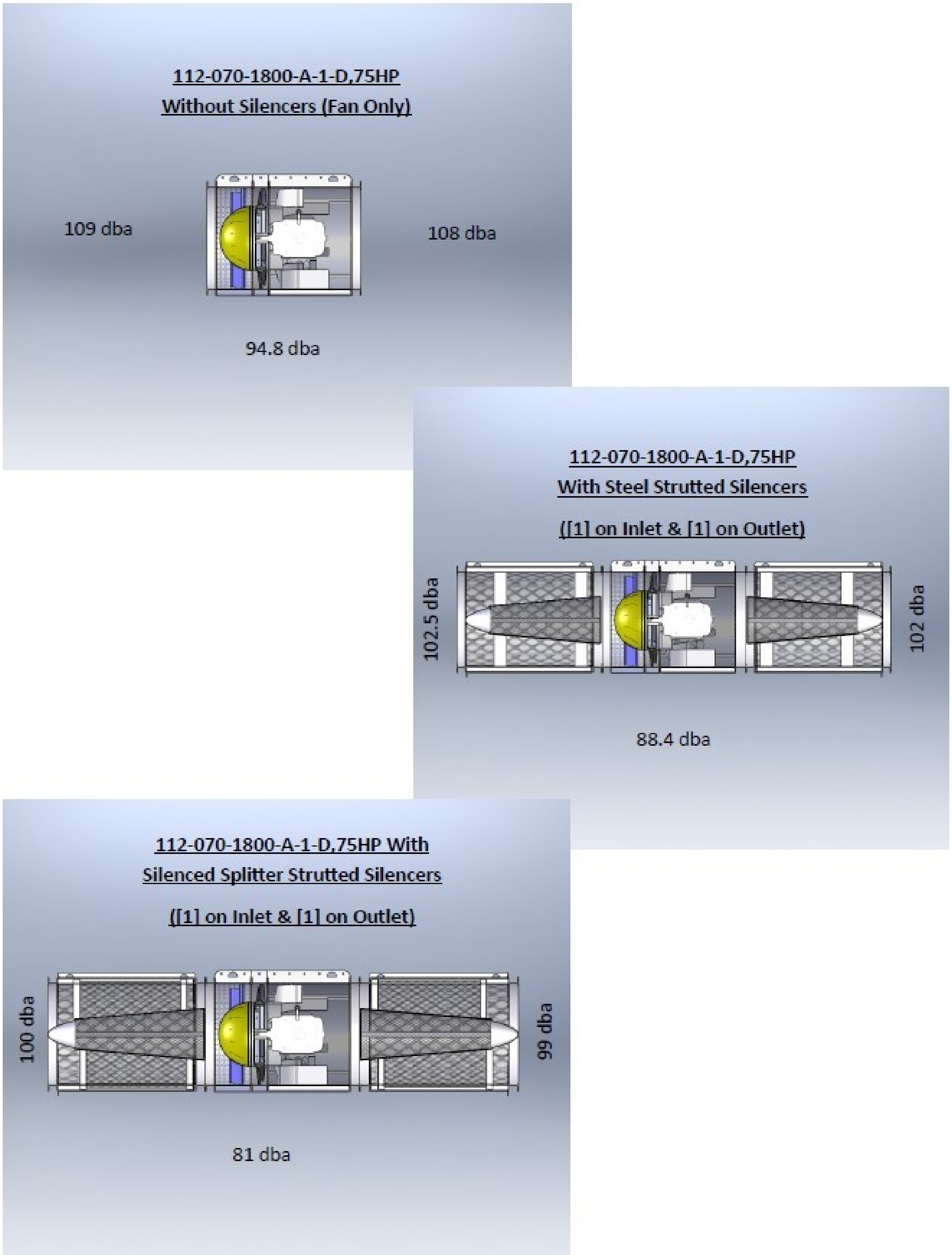



**Figure 2.6: Noise Reduction Provided by Silencers (Spendrup Fan Co.).**

# 3.0 Geologic Design Factors

While there is a tendency to focus on environmental factors that exist above the surface at any particular mineral property, it is important to consider that the conditions below the ground will have an equally significant impact on a subterranean mine.

Exploration drilling will reveal details of the mineralization, strata temperatures and morphology of the orebody and host rock that can all affect the underground environment negatively and require mitigation. In this manner, the ventilation system design may depend, at least partially, on the vagaries of geology.

# 3.1 Geothermal Conditions

Of all geologic conditions that affect the ventilation system of a mine, perhaps none are so significant as the *virgin rock temperature* or VRT of the host rock. The in-situ strata temperature of a rock mass depends on a range of factors that include the thermal conductivity of the strata, the thickness of the earth's crust, the presence of water and size of any fractures present.

For future projects (e.g., feasibility studies, etc.), the VRT can be determined from down-hole drill data logs. It should be noted, however; that there is an area close to the surface where strata temperatures are greatly influenced by the proximity to the ambient environment (which acts as a heat sink) and can lead to temperature anomalies in extrapolation.

In existing mines, VRT can be measured directly by placing thermocouple probes in drill holes at the active mining face. Borehole temperatures peak approximately one shift after blasting, after which time the strata will begin to cool as a result of the heat transfer to the surrounding air (Duckworth, 1999).

Often, it is desirable to extrapolate known temperature data (VRT) to areas that have yet to be excavated or otherwise measured (i.e., future planning exercises). For this purpose, the *Geothermal Gradient* (°C/100m) or *Geothermal Step* (m/°C) are used. Geothermal Gradient, once determined, can then be utilized to project the temperature of the rock strata at any known depth.

Geothermal Gradient is calculated by measuring VRT in two places where the elevation is known to a high degree of accuracy (i.e., surveyed). The slope of the line between the two points on a graph of temperature versus depth will then give the Geothermal Gradient or Step, depending on the alignment of axes.

It is important to remember that VRT and Geothermal Gradient can vary greatly even in a relatively small geographic area, so any projections of this type should be verified by actual measurements as soon as reasonably possible in the design process.

Water inflows also represent significant sources of geothermal heat for many mines. Depending on the temperature of the groundwater, and the rate of entry to the mine workings, groundwater may, or may not be a limiting factor with regard to the mine heat load.

Any comprehensive ventilation design must account for the total heat load of the mine, and if that heat contribution is significant, then a complete climatic analysis of the mine should be performed by utilizing one of the commercially available software packages designed for this purpose.

An accurate climatic model will allow for the accurate determination of the amount of cooling that will be required to meet the project design criteria, and allow for greater efficiency in the design of any mechanical refrigeration system that is required.

# 3.1 Geothermal Conditions

Of all geologic conditions that affect the ventilation system of a mine, perhaps none are so significant as the *virgin rock temperature* or VRT of the host rock. The in-situ strata temperature of a rock mass depends on a range of factors that include the thermal conductivity of the strata, the thickness of the earth's crust, the presence of water and size of any fractures present.

For future projects (e.g., feasibility studies, etc.), the VRT can be determined from down-hole drill data logs. It should be noted, however; that there is an area close to the surface where strata temperatures are greatly influenced by the proximity to the ambient environment (which acts as a heat sink) and can lead to temperature anomalies in extrapolation.

In existing mines, VRT can be measured directly by placing thermocouple probes in drill holes at the active mining face. Borehole temperatures peak approximately one shift after blasting, after which time the strata will begin to cool as a result of the heat transfer to the surrounding air (Duckworth, 1999).

Often, it is desirable to extrapolate known temperature data (VRT) to areas that have yet to be excavated or otherwise measured (i.e., future planning exercises). For this purpose, the *Geothermal Gradient* (°C/100m) or *Geothermal Step* (m/°C) are used. Geothermal Gradient, once determined, can then be utilized to project the temperature of the rock strata at any known depth.

Geothermal Gradient is calculated by measuring VRT in two places where the elevation is known to a high degree of accuracy (i.e., surveyed). The slope of the line between the two points on a graph of temperature versus depth will then give the Geothermal Gradient or Step, depending on the alignment of axes.

It is important to remember that VRT and Geothermal Gradient can vary greatly even in a relatively small geographic area, so any projections of this type should be verified by actual measurements as soon as reasonably possible in the design process.

Water inflows also represent significant sources of geothermal heat for many mines. Depending on the temperature of the groundwater, and the rate of entry to the mine workings, groundwater may, or may not be a limiting factor with regard to the mine heat load.

Any comprehensive ventilation design must account for the total heat load of the mine, and if that heat contribution is significant, then a complete climatic analysis of the mine should be performed by utilizing one of the commercially available software packages designed for this purpose.

An accurate climatic model will allow for the accurate determination of the amount of cooling that will be required to meet the project design criteria, and allow for greater efficiency in the design of any mechanical refrigeration system that is required.

# 3.2 Geotechnical Conditions

Geotechnical instability has the potential to occur in any mine tunnel or opening (including verticals shafts and raises) where the air temperature will cross the threshold between liquid and solid water. When water is present in rock strata, either within pore space or flowing via fractures, faults, slips, etc., the rock may be damaged by repetitive freezing and thawing of the water owing to temperature fluctuations that are either natural (e.g., seasonal, diurnal) or induced (e.g., auto-compression, diesel equipment, etc.).

The expansion and contraction of the strata, especially along cracks and faults can significantly damage the rock, and eventually lead to ground control failures that may range from spalling to complete collapse. The stability of the rock ultimately depends on a wide range of factors that includes ambient temperature, precipitation, rock type, RMI, permeability among others.

The potential instability that arises from repetitive freeze/thaw cycles is most commonly controlled by bulk air heating. Bulk air heaters located at the intake portals and shafts heat the air sufficiently to ensure that any water present in the rock strata does not freeze and expand, thus protecting the integrity of the excavation.

Mine openings susceptible to this type of instability may also be protected by a variety of liners, and/or sealants.

In some cases, such as when mine openings are excavated through areas of permafrost, the integrity of the ground is protected by maintaining the temperature of the air (and consequently the ground) consistently below the freezing temperature of water. In areas of permafrost, it may become necessary to chill the air with refrigeration or use a ground-freezing brine injection system to protect the integrity of the mine openings.

Some northern mines require bulk air heating in the winter months and bulk air cooling in the Summer. This condition represents a unique challenge to the ventilation engineer, who must consider how to balance the need to protect the underground workforce from dangerous temperature/air velocity combinations and the need to protect the permafrost from becoming unstable due to above-freezing temperatures. Drift insulation may also be used, such as proprietary shotcretes or spray-on polyurethane products. In any case, an unplanned failure of the ground support systems represents a dangerous condition that can result in significant losses for the mine if not mitigated.

In the summer of 1983, partial melting of the permafrost at the Polaris Mine led to an unplanned mine closure. The mine was eventually re-opened after the installation of several bulk air refrigeration units located at the main mine intakes (Cominco, Ltd., 1984).

## Case Study: Polaris Mine

Operating from 1981 to 2002, the Polaris Mine was the world's northernmost base-metal mine. This lead-zinc mine was located approximately 700 miles north of the Arctic Circle on Little Cornwallis Island in the province of Nunavut, Canada. The underground mine itself was almost entirely contained within an area of permafrost, which necessitated several innovative techniques with regard to the mine environment, and which are still directly applicable to many operating mines, as well as any future mines planned in Arctic/Antarctic regions.

Ambient temperatures at the surface of the Polaris Mine ranged from approximately -58 degrees Fahrenheit in winter to 59 degrees Fahrenheit in summer, with permafrost that extended to a depth of 1,400 feet below the surface. With the entirety of the mine development located within the permafrost, it was necessary to prevent the ground from thawing at all times. During winter, the ambient temperatures were sufficient to prevent thawing of the drift walls, but in summer, the elevated surface temperatures coupled with the heat from the mine equipment (e.g., trucks, loaders, etc.) necessitated the refrigeration of intake air.

Ultimately, four bulk air chilling units were installed to ensure that the mine air never rose above freezing, even with the significant heat-load produced by the mine equipment during full production. The sub-freezing conditions protected the integrity of the mine openings, and offered a secondary benefit; eliminating the need for cemented backfill (which would have added too much heat) and replacing it with low-cost rock mixed with water and frozen solid.

Today, the lessons learned during the development and operation of the Polaris Mine are applicable to anyone who operates a mine in areas of permafrost (even in areas where only a portion of the mine development is located in permafrost), or when freezing of the ground is required for geotechnical stability or groundwater control.

# 3.3 Geologic Hazards

Geologic hazards that affect the design of underground ventilation systems include host or ore rock that contains gases that can negatively impact the safety of an operation such as radon, hydrogen sulfide, or methane, or minerals that constitute a health hazard (e.g., asbestos, silica, etc.).

Gaseous contaminants present in an underground reservoir are generally controlled through dilution via the ventilation system or by draining the gases through pre-drilled wells that bleed the gases from the rock prior to excavation.

The amount of airflow required to dilute the contaminant gas(es) can be calculated based upon the expected rates of inflow to the mine entries and the recommended (or legislated) safe or accepted level of the contaminant(s) in the airstream.

Particulates such as dust, silica dust and asbestos are controlled through dilution and removal via the ventilation system. Achieving the correct range of airflow velocity is important, whereby existing dust can be removed from the area, but additional dust is not created by the airstream. This optimum velocity range for dust control exists between 1 m/s and 3 m/s.

If the primary ventilation system is insufficient to mitigate the hazard, then additional controls will have to be implemented. Water sprays, and wet and dry dust scrubbers have proven effective at removing or reducing dust from underground environments.

If radiation or radioactive contaminants are encountered, the recommended strategy for mitigation is total avoidance of the hazard. This is practically accomplished by designing a ventilation system whereby fresh air is first passed over the workers, then over the hazard and finally out of the mine (or at least directly into the exhaust system). The direction of air leakage within the system should always flow from the mine intake to the exhaust. This is achieved through the manipulation of differential pressures within the ventilation circuit, via fans and regulators.

# 4.0 Strategic Design Factors

Strategic design factors that will affect the ventilation system design for metal and non-metal mines include those that result from choices made by the mine design team, e.g., mining method, size and location of underground shops and other facilities, the size and type of mechanized equipment, etc. Often, these are the variables that have the greatest impact on the design of the ventilation system.

# 4.1 Mining Method

The choice of a mining method is somewhat dependent on the size and morphology of a given ore deposit; however, almost all economically feasible mineral properties will have some options available when it comes to methods of extraction, and the choice of mining method is a critical parameter in the design of a mine and mine ventilation system.

Variations in the size and scope of an orebody will naturally favor some methods of extraction than others, and consequently lead to great differences in both the total amount of airflow required and also the ventilation system infrastructure required to distribute that airflow effectively.

Although many mines share common characteristics (e.g., tonnage rates, mining methods, or ore type), no two mines are entirely alike, and the design of their ventilation systems should be likewise adjusted to account for their differences.

The individual components of the mine and infrastructure should be carefully considered when designing a ventilation system and each factor identified. Consider how the design of a ventilation system would change if a mine utilizes a conveyor level and central shaft to haul ore compared to one that accomplishes its mineral transport with a large fleet of diesel trucks that haul ore up a series of underground ramps to reach the surface. Are there many active production zones at any given time or just a few? Is the mine located near the surface, or deep within the ground? What is the expected mine life? All of these questions will need to be answered by the ventilation engineer during the design phase of a project.

The ventilation system design for individual mining methods that will be discussed in detail include:

### Block Caving

Block caving is the most complex and infrastructure intensive mining method, which requires extensive ventilation infrastructure, and often quantity. Generally, block caves have entire levels developed exclusively for ventilation, and many block cave mines will have more than one. Significant ventilation is required for the production and haulage levels, as well as ramps and shops. Of course, block caves also require vast mobile equipment fleets that also must be considered. Because of their high tonnage rates, and a need to keep the cave moving, or active, many block cave mines are time-sensitive rather than cost-sensitive when it comes to mine planning. This can change the economic considerations of a future planning exercise, and it is an important consideration when designing ventilation systems for block cave mines.

Figure 4.1 shows a screenshot of a ventilation model for a block cave mine.

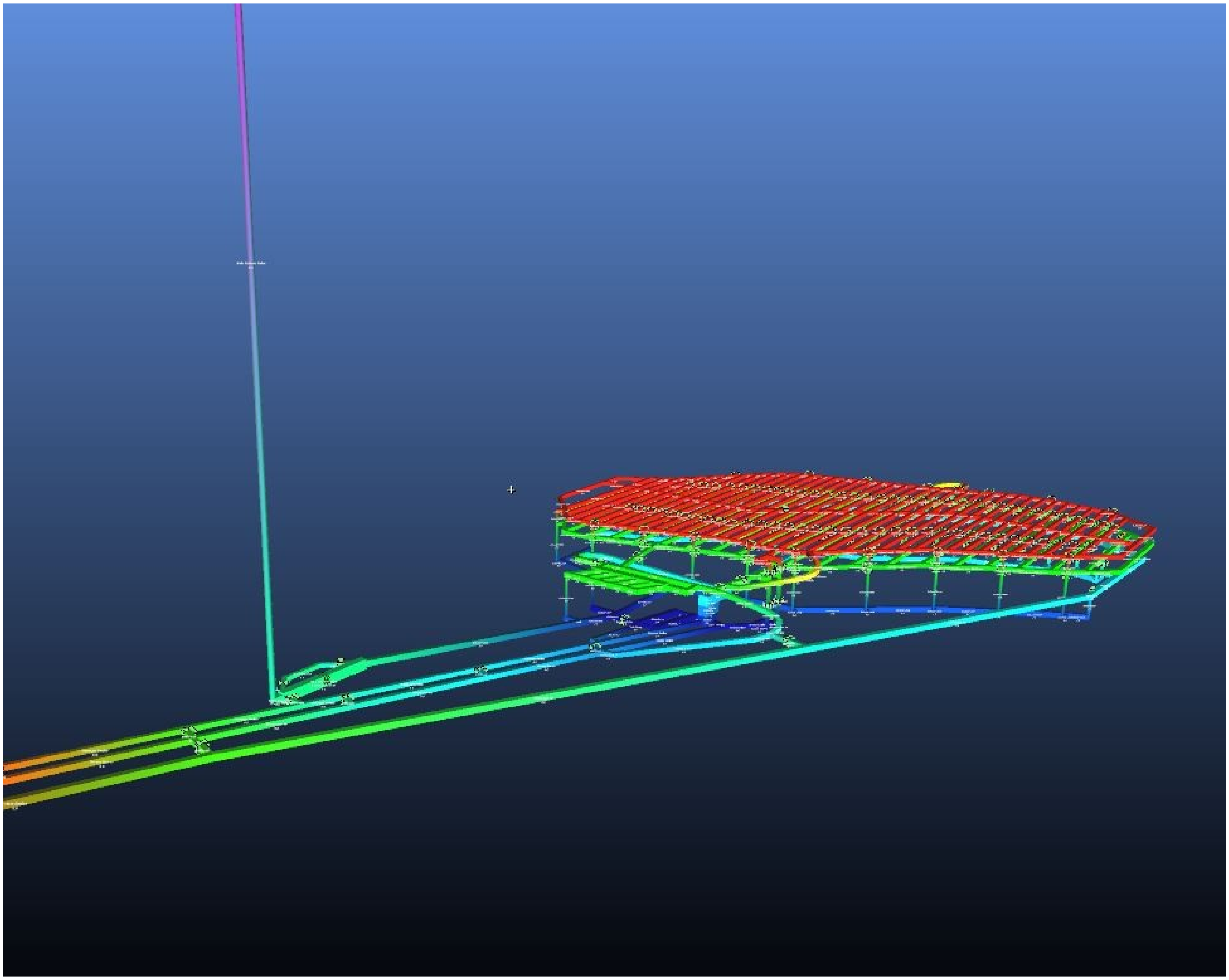


**Figure 4.1: Ventilation model of a block cave mine.**

## Sub-level Caving

Sub-level caving is a commonly-utilized mining method when conditions are not amenable to block caving, but high production rates are required. The mine infrastructure may consist of many ramps and sub-levels in combination, that are often ventilated with intake and return raises.

Figure 4.2 shows a screenshot of a ventilation model for sub-level caving mine.

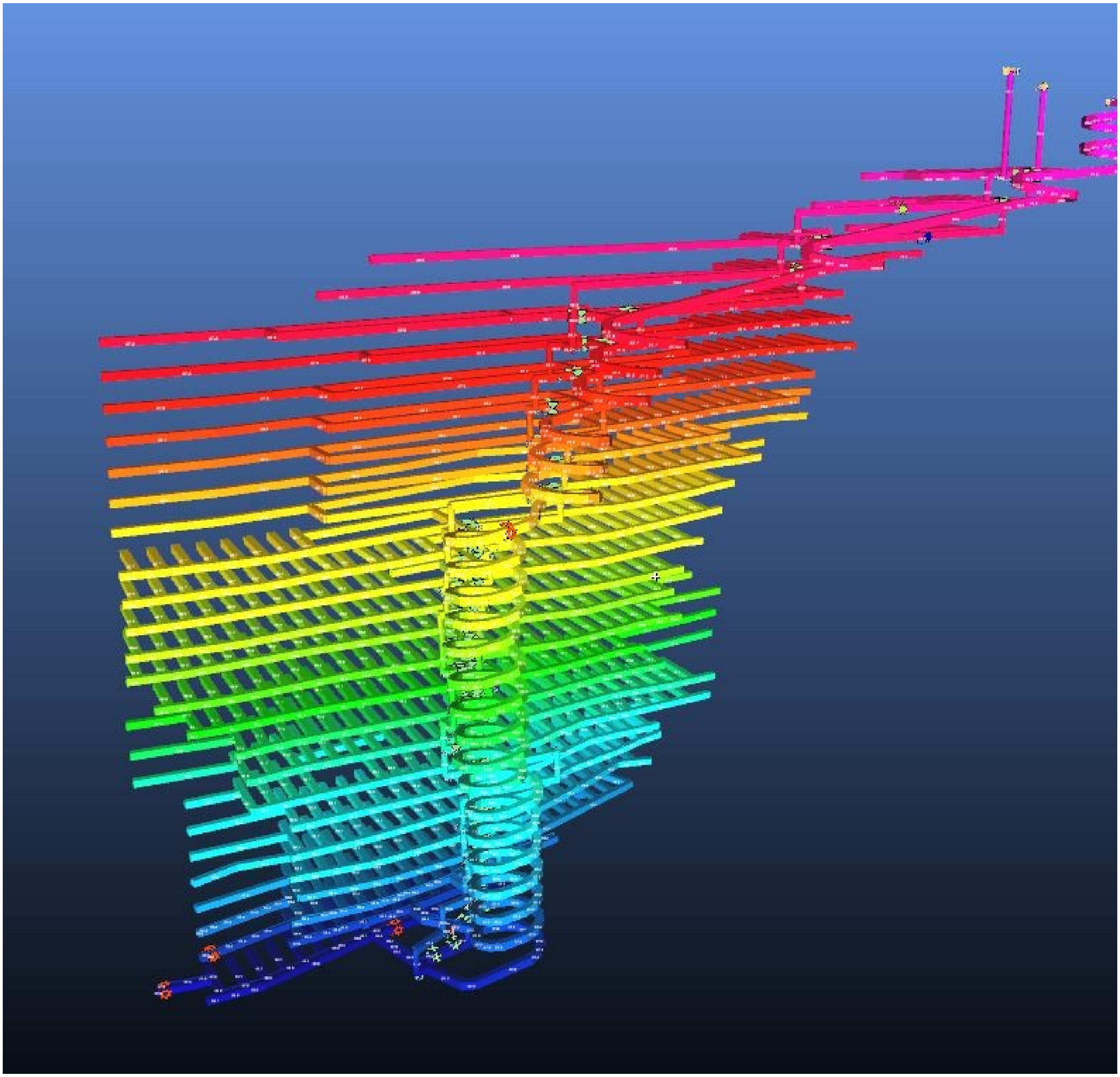


**Figure 4.2: Ventilation model of sublevel caving mine.**

## Cut and Fill Stoping

Cut and fill mines are often used to mine narrow-vein deposits that dip steeply along their extents. A ramp is developed close to the orebody, which is then accessed through a series of perpendicular tunnels. After completion of production, the mined-out tunnel is then filled, often with a combination of waste-rock and cement, or backfill. All infrastructure in a cut and fill mine must be developed in the foot wall in order to ensure its integrity throughout the Life of Mine. Although it is sometimes possible to develop intake and exhaust raises in these types of mines, the production levels of cut and fill mines are most commonly ventilated with auxiliary ducts that are run from the ramp to the active faces (usually two or more per level).

Figure 4.3 shows a screenshot of a ventilation model for a cut and fill mine.

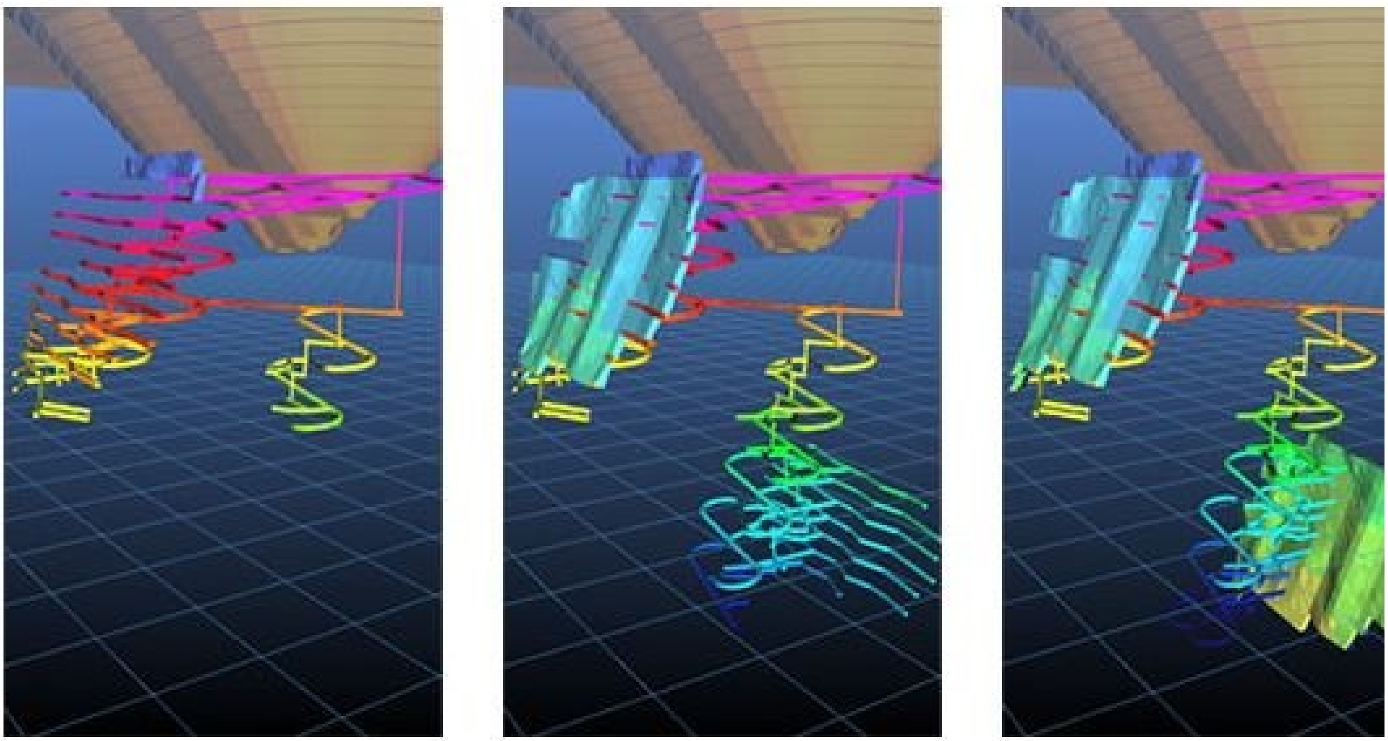


**Figure 4.3: Ventilation model of a cut and fill mine.**

### Room and Pillar

Room and pillar, or bord and pillar mining methods have been used in many mines with tabular, or flat deposits, including salt, stone and base metals. The rooms may be excavated mechanically with mining machines, or via a more traditional drill, blast, muck cycle. Room and pillar mines generally require a large volume of airflow in order to maintain minimum velocities in what amounts to many parallel entries. The number of bulkheads, regulators and overcasts (or cross-overs) is large, which also results in increased leakage potential. Proper construction and maintenance of ventilation controls is essential in these environments, and differential pressures should be kept as low as possible in order to minimize leakage. This is often accomplished through the use of push-pull ventilation systems. As a result of these low pressures, face ventilation may be accomplished via brattices or curtains, rather than with auxiliary fans and ducts.

Figure 4.4 shows a screenshot of a ventilation model for a room and pillar mine.

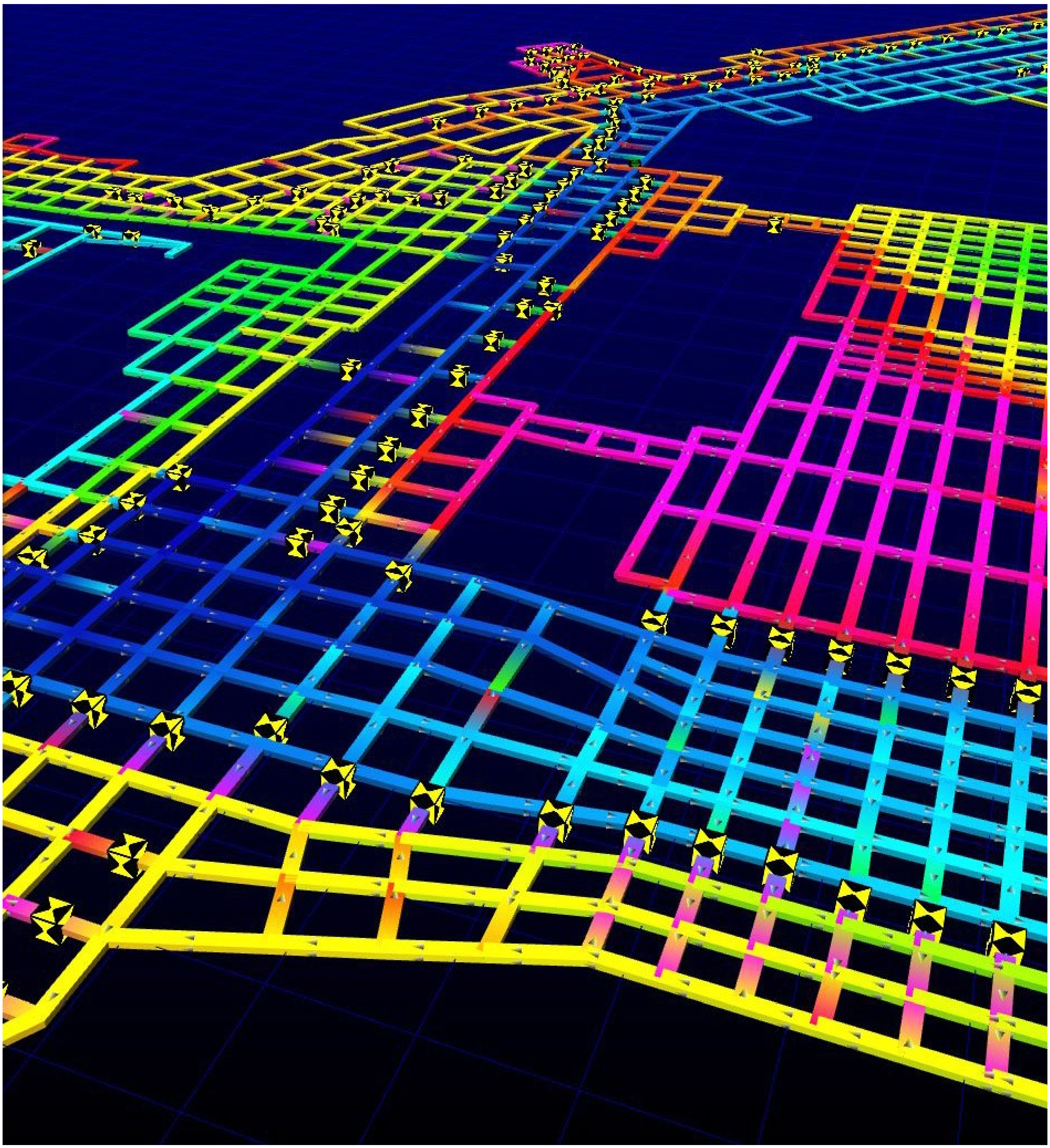


**Figure 4.4: Ventilation model of a room and pillar mine.**

# 4.2 Conceptual Design

Once the geographic and geologic parameters have been identified, and the choice of an appropriate mining method is made, then the *conceptual design* for the ventilation system can be considered. Generally speaking, there are many factors affecting ventilation system design that can be considered to fall into the category of conceptual design factors.

These factors include the choice of fan configuration (i.e., blowing, exhausting, or push-pull), the configuration of transport and haulage routes, fixed facilities and surface connections and the location and type of ventilation controls such as doors, walls and regulators.

The process of conceptual design is often iterative, and it will be necessary to work closely with other designers and/or departments during this phase of the process. For example, if the ventilation engineer is not aware of the geotechnical conditions, they may call for a ventilation raise that will not be supported by the rock type/strength. Alternatively, a drift may be sized appropriately for optimal airflow velocity, but not large enough to accommodate the equipment necessary to meet the production goals of the mine.

It is this type of interaction and compromise between the competing interests and components of the mine design team that is critical to the design process and will ensure that the ventilation system design will adequately support the project. It is important to remember that the ventilation system is there to support the mine operations and not the other way around.

Because mine ventilation is perceived as a necessity in order to protect the health and safety of the workforce, it is frequently considered to be a pure "cost" of mining operations without regard to the contribution to production and productivity (this in spite of many published reports that demonstrate the relationship between healthy and safe workers and increased productivity and show equal correlation between high rates of accidents with higher operating costs).

This misconception often results in a series of compromises with the design of the ventilation system, between the risk to the workforce mitigated by the ventilation system and the cost required to implement the design or components of it. Although compromise is necessary in any such large and complex design, great care should be taken before any measure to protect the underground workforce is reduced or removed.

The ventilation of active mine workings in series (whereby air travels first over one area, then over another) should be avoided whenever possible. If it is not possible to avoid ventilating multiple headings in series, additional measures to ensure the safety of the workforce should be implemented.

Dr. Rick Brake of Mine Ventilation Australia recommends the following four controls be implemented whenever series ventilation of mine workings occurs:

1. A second means of escape, each with its own independent fresh air supply that is pressurized relative to the workings (such that air flows from the secondary egress into the mine workings). This escape path should be kept free of any inflammable materials.
2. Secure and self-contained refuge chambers should be located in close proximity to the working areas.

3. Workers should be supplied with Self Contained Self Rescuers (SCSRs) that are sized appropriately to the conditions and escape routes and theses SCSRs should be carried underground at all times. Escape drills should be conducted regularly and under those conditions likely to be experienced during an actual emergency (e.g., smoke, heat, etc.).
4. The mine should install and maintain a system of early-warning sensors to detect combustion (e.g., POC, CO, etc) as well as utilizing an advanced system designed to notify and inform underground workers in the event of an emergency (i.e., PED).

Even these conditions may not be considered sufficient protection in some cases where active mine workings are ventilated in series, but they should be considered a minimum requirement for the protection of the workforce in such conditions.

Another potentially hazardous (although altogether too common) ventilation system design flaw exists in mines that utilize the primary haulage ramp as the principal intake pathway for a mine. This type of system has been observed in many small metal and non-metal mines that have chosen this configuration as a way to minimize mine infrastructure required and the resulting cost of development. In modern mechanized mines with a large diesel fleets, the fresh air source for the mine is thereby contaminated along its entire source. Furthermore, the single greatest source of fire in the mine (in terms of fuel source and probability) is located in the principal intake. Systems of this type should be avoided if at all possible, and should only be considered if measures such as those suggested by Dr. Brake for series ventilation are also implemented.

Although major underground fires are considered low probability events, a Niosh/CDC report states that they are to blame for over 90% of the mining disasters (accidents resulting the loss of five or more lives at once) in metal and non-metal mines in the past 150 years.

Another important concept that guides underground ventilation system design is the *ALARA Principle*. ALARA stands for “As low as reasonably achievable”, and is sometimes described as ALARP, or “as low as reasonable possible”. In all mine systems design, ALARA dictates that all risks to the workforce be kept as low as reasonably achievable. Following the ALARA Principle in ventilation system design often results in standards and practices that exceed those that are required by any applicable regulations and/or governmental oversight.

Finally, the question of automation and control systems should be addressed. What systems will be installed? What level of control over the ventilation system will rest the technical staff of the mine versus the portion that will be controlled automatically via a central computer? Several companies and manufacturers now offer both automated controls and monitors as well as “smart” system controllers designed to make complex decisions regarding conditions and airflow distribution throughout the mine in real time. Systems of this type, often referred to as *Ventilation On Demand* or VOD have demonstrated great potential for improving the efficiency of the ventilation system overall and minimizing any required operating costs; however, they are not without some risk associated with actual environmental conditions underground. Any installed VOD systems should be extensively researched, designed and maintained by expert technical staff. Additional information regarding these types of systems can be found the “Ventilation System Automation and Control” course in this series of educational modules.

# 4.3 Equipment Selection

The Selection of mechanized equipment for a new mine has a great impact on the demands for ventilation system. The size and type of equipment required to support the mine's production and development schedules will dictate how much airflow is needed to dilute the gaseous and particulate components of the exhaust, and to remove any heat that is generated above certain, prescribed levels. This selection process may also govern the size of certain critical ventilation infrastructure, such as access ramps and drifts, or main mine levels. The process is essentially the same when adding new equipment to the mine, even if the size of existing ventilation infrastructure may not be altered.

Generally, mining equipment is selected by purchasers based on its ability to meet the production and development schedules required by mine economic model(s) and chosen for its lowest cost. Often, the impacts to the ventilation system are not considered until the equipment is purchased, and operating (if at all). Often, this leads to significantly higher operating costs, when the ventilation system design is considered, as the emissions profiles for various engine packages can vary greatly (even among packages with the same power output). Operating equipment will also add additional heat and dust into the underground environment, which will add to the demands on the ventilation system- problematic if the system is already operating at or near capacity, as is often the case.

Historically, trends in diesel equipment use in underground mines show that engine size (power), total installed diesel power mine-wide, and the size (physical dimensions) of equipment are all increasing relative to time. In addition to needing additional airflow for the reduction of tailpipe emissions, this new, larger equipment often requires a larger drift in which to operate. This in turn, causes additional demand on the ventilation system based on the minimum velocity required to physically remove contaminants from the vicinity, and in some cases exceed the quantity required for dilution of exhaust gases/particulates.

The determination of the airflow required for a diesel equipment fleet may be calculated in a number of ways, including but not limited to regulatory compliance, nameplate ventilation or direct engine testing and other empirical methods.

As we have already discussed, diesel equipment operating in underground mines produces gaseous and particulate contaminants from the tailpipe as well as heat and mineral dust that must be considered by the ventilation engineer or ventilation system designer. In light of this, it is recommended that all four types of contaminants be considered when determining airflow quantity required for diesel equipment. It should also be noted, that compliance with all regulatory requirements must be maintained at all times, regardless of the method(s) used.

Table 4.1 gives regulatory requirements for selected countries/regions around the world.

**Table 4.1: Selected regulatory requirements for controlling diesel emissions.**

| **Location** | Statutory Ventilation Rate(s) |
| --- | --- |

| | |
|---|---|
| Australia | 0.06 $m^3/s$ per kW minimum |
| Canada | Varies by province from 0.045 – 0.092 $m^3/s$ per kW minimum – most commonly 0.06 $m^3/s$ per kW |
| Chile | 2.83 $m^3/min$ per effective brake horsepower minimum (0.063 $m^3/s$ per kW equivalent) |
| China | 0.067 $m^3/s$ per kW |
| South Africa | 0.063 $m^3/s$ per kW minimum (based on "best practice") |
| USA | Based on MSHA certificate – ventilation required to dilute contaminants to specified levels at the tailpipe |

The above table raises several questions regarding the operation of diesel equipment underground. For example, how are the contaminants produced by an LHD operating in Australia different than those produced by the same loader operating in Canada? What about the contaminants from a loader operating in the Copiapó region of Chile (at 1,000 mASL) compared to one operating in the Andes region of that country (4,000 mASL)?

Despite these and other issues, this is the most commonly used method for determining airflow requirements for underground mines around the world.

In most cases, these airflow multipliers were based upon empirically-derived relationships between diesel emissions and airflow. While this method often gives the most reliable data, care should be taken that the sufficient data points are included in the calculations, and that similar input parameters (vis a vis equipment age, type, use and location) exist in order to make the comparisons valid. In light of the potentially large number of data points necessary to derive accurate values, this methodology is not available to everyone. Empirically-derived multipliers for determining the required airflow for diesel-powered equipment typically ranges from 100 – 150 cfm/bhp (Stinnette and DeSouza, 2013).

In some countries (e.g., Canada, U.S.), regulators test the tailpipe emissions of approved diesel engines directly, to determine the airflow required to dilute these contaminants to the required TLVs. In Canada for example, an Exhaust Quality Index (EQI) has been developed, based on the following relationships:

$$EQI = \frac{CO}{50} + \frac{NO}{25} + \frac{DPM}{2} + 1.5\left[\frac{SO_2}{3} + \frac{DPM}{2}\right] + 1.2\left[\frac{NO_2}{3} + \frac{DPM}{2}\right]$$

Where DPM is given in milligrams per cubic meter and the gas concentrations represent parts per million present in the raw exhaust. During an 18-mode engine test designed to replicate the engine loading of an operating piece of equipment, the flow rate required to produce an EQI of "3" is calculated.

In the U.S., MSHA determines the required ventilation rate to dilute the gaseous emissions to 50 ppm CO, 5,000 ppm $CO_2$, 25 ppm NO and 5 ppm $NO_2$ at each of point of an 8-mode engine test. At particulate index (PI) is also calculated, which constitutes the quantity of airflow needed to produce 1 milligram per cubic meter.

Although these methods of direct testing provide a specific and repeatable means for determining the airflow required for a diesel engine, they still do not consider the heat or dust that will be generated by the equipment when it is in use in an actual underground mine. It should also be noted that these ventilation rates based on direct measurements, are usually lower than those derived empirically or specified in mining regulations, making their value somewhat limited except as a comparison between two or more engines directly.

In 2013, a new method for determining the volume of airflow required for a piece (or fleet) of diesel-powered equipment was suggested by Stinnette and DeSouza at Queen's University. This method involves the determination of the airflow required for each type of contaminant produced (i.e., exhaust gases, particulates, heat and dust) and then selecting the greatest airflow requirement. This methodology is demonstrated on Figure 4.6.

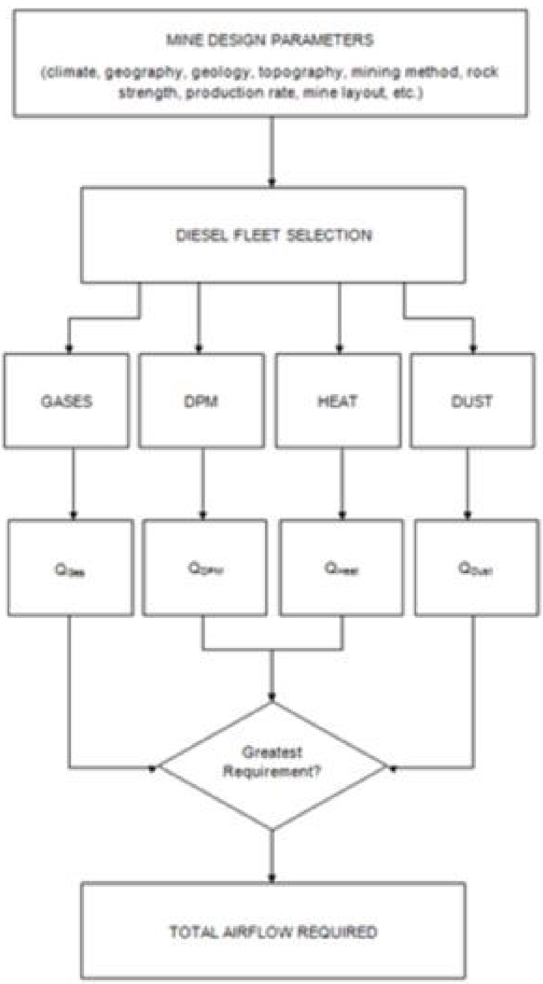



**Figure 4.6: Determining airflow quantities for diesel-powered equipment.**

The gaseous and particulate emissions from the vehicle can be obtained from direct engine testing where available, or else determined from empirical multipliers. Recommended airflow multipliers based on the age/classification of the equipment are given in Table 4.2.

**Table 4.2: Ventilation Rates for MSHA approved engines (Haney, 2012).**

| EPA Tier | Gaseous Vent (cfm/hp) | PI (cfm/hp) | PI X 5 (cfm/hp) |
|---|---|---|---|
| Non EPA Compliant < 73 kW (99 hp) | (79 ± 90) | (188 ± 139) | (942 ± 693) |
| Non EPA Compliant > 73 kW (99 hp) | (60 ± 12) | (94 ± 38) | (324 ± 120) |
| Tier I/II < 73 kW (99 hp) | (60 ± 15)* | (65 ± 24) | (324 ± 120) |
| Tier I/II > 73 kW (99 hp) | (55 ± 12)* | (31 ± 15) | (156 ± 74) |
| Tier III < 73 kW (99 hp) | (50 ± 7)** | (44 ± 23) | (219 ± 113) |
| Tier III > 73 kW (99 hp) | (39 ± 5)** | (39 ± 14) | (194 ± 72) |
| Tier IV | (39 ± 5)** | (3.2)*** | (16)*** |

*Based on NO **Based on $CO_2$ ***Based on a PI of 0.01 gm/hp-hr

The airflow required to dilute and remove mineral dust generated by the equipment in operation is based on a minimum recommended airflow velocity of 200 fpm (1 m/s). This value is based upon the minimum airflow velocity required to minimize the entrainment of dust while diluting respirable dust (McPherson, 2009) and achieving the maximum penetration depth into dead-end headings (Rawlins and Hardin, 2005). Figure 4.7 shows the relationship between relative dust concentration and airflow velocity. Figure

4.8 gives the maximum penetration depth into a blind headings developed at 10 ft by 10 ft and 13 ft by 13 ft versus airstream velocity.

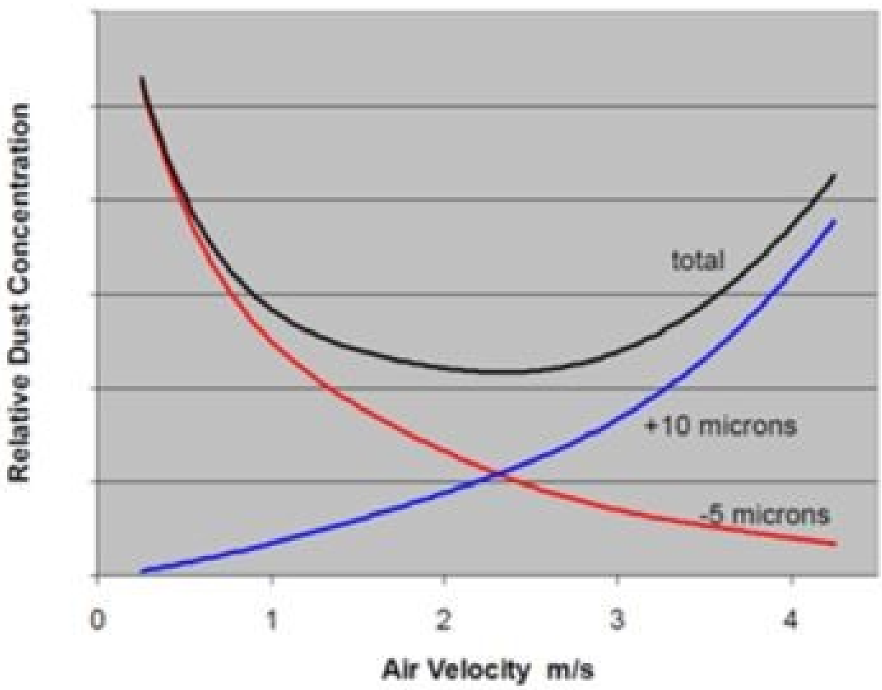



**Figure 4.7: Relative dust concentration vs. air velocity (McPherson).**

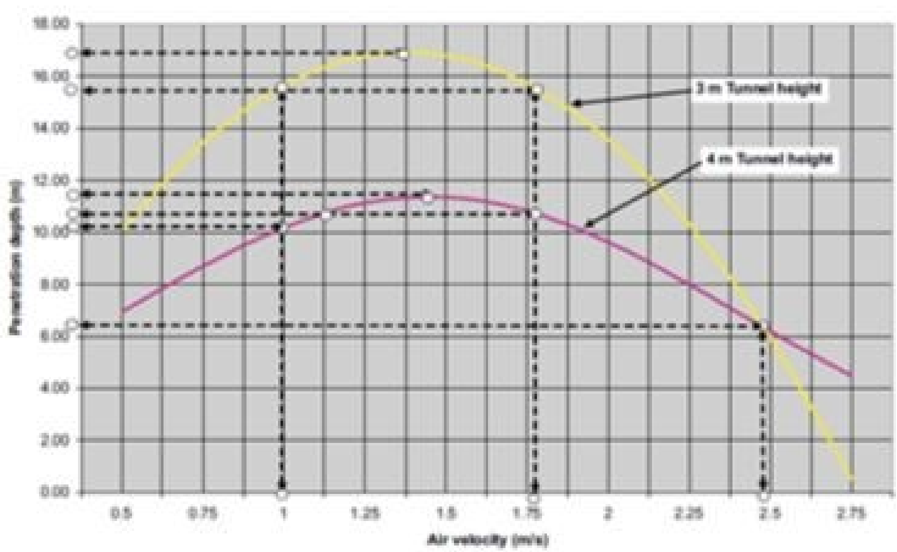



**Figure 4.8: Penetration depth in blind headings for various airstream velocities (Rawlins and Hardin).**

It should be noted that in cases where the generation of mineral dust in the driver in determining the required airflow for the diesel equipment fleet, that quantity may be reduced if other means of dust control (e.g., water sprays, collectors, etc.) are implemented.

A diesel engine can be expected to produce as much as three times as much heat as the amount of usable work in bhp or kW. This occurs as radiative heat losses from the machine itself (radiator, exhaust pipe, etc.), heat of the gaseous tailpipe emissions, and frictional heat generated by the bucket, tires, etc. Further compounding this process is the fact that between three and ten liters of water are generated for each liter of diesel that is combusted. Figure 4.9 shows the breakdown of heat produced by an operating piece of diesel equipment relative to the work performed.

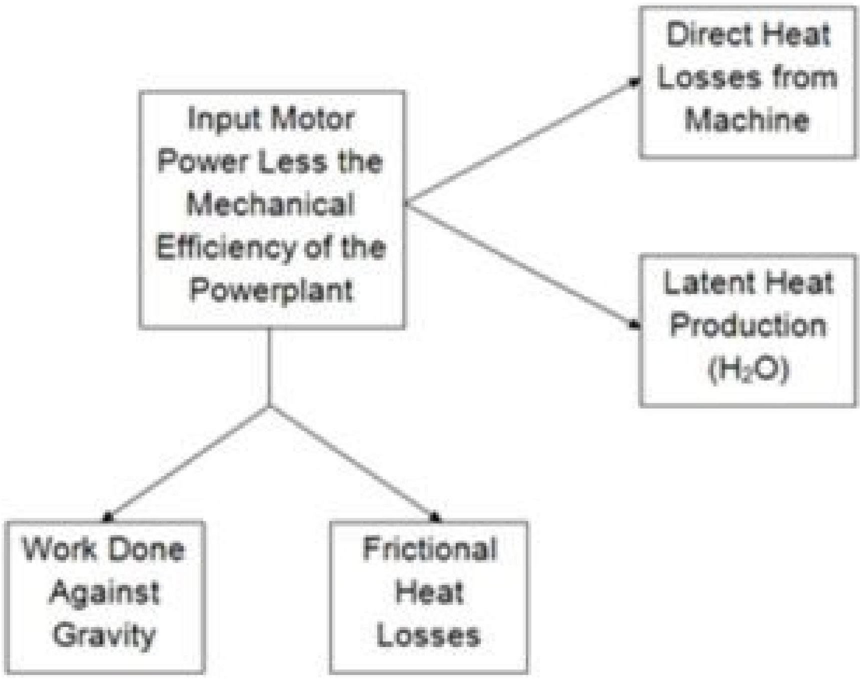



**Figure 4.9: Components of Work and Heat produced by operating diesel equipment.**

Heat calculations are generally performed using the SI system of units, owing to the great simplification of the equations/calculations involved. The conversion of engine power and airflow rates between SI and Imperial systems of units are easily and quickly accomplished.

Determining the airflow required by a given piece of diesel equipment based on its expected heat production can be achieved by first calculating the Total Heat:

$$Q_t = \frac{f_c \times C_{diesel}}{3600}$$

where: $Q_T$ = total heat (kW)

$f_c$ = fuel consumption (litres/hr)

$C_{diesel}$ = heat content of diesel (kJ/litre)

Next the latent heat production of the equipment is determined:

$$Q_l = \frac{V_{H2O} \times l_{H2O}}{3600}$$

where: $Q_l$ = latent heat (kW)

$V_{H2O}$ = volume of water production (litres/hr)

$l$ = latent heat of vaporisation of water (kJ/kg)

The difference between the Total Heat and the Latent Heat represents the Sensible Heat:

$$Q_S = Q_T - Q_l$$

where: $Q_S$ = sensible heat (kW)

$Q_T$ = total heat (kW)

$Q_l$ = latent heat (kW)

The associated temperature rise in the ambient air across the machine is a function of the flow rate of air:

$$m_{air} = \frac{Q_S}{\Delta T \times C_p}$$

where: $\dot{m}_{air}$ = mass flow rate of air (kg/s)

$\Delta T$ = temperature change (K)

$Q_S$ = sensible heat (kW)

$C_p$ = specific heat of dry air (kJ/kgK)

In order to calculate the ventilation rate necessary to limit the temperature increase across the machine to a certain maximum value, or to ensure that conditions do not reach a given temperature limit, the mass flow rate of air can be calculated . Note that the mass flow rate of air should be converted to a volume flow rate using the air density in order to compare it with other flow-rate criteria:

$$VR = \frac{v_{air}}{P_{machine}}$$

where: $VR$ = ventilation rate ($m^3$/s per kW)

$v_{air}$ = volume of air required ($m^3$/s)

$P_{machine}$ = machine power (kW)

Table 4.3 shows a comparison of the ventilation rates calculated using various methods for a typical LHD approved for use in North-American underground mines. In the case of heat, the temperature rise across the machine was limited to 36 degrees Fahrenheit or 20 degrees Centigrade.

**Table 4.3: Comparison methods for calculating the total airflow required for a diesel Loader.**

| Method of determining Airflow | Total Airflow ($m^3/s$) | Ventilation Rate ($m^3/s$ per kW) | % of Greatest (%) |
|---|---|---|---|
| Direct Engine Testing* | 5.9 | 0.021 | 18% |
| Empirical Derivation | 18.0 | 0.063 | 55% |
| Proposed Method: Gases | 8.0 | 0.028 | 25% |
| Proposed Method: DPM | 3.1 | 0.011 | 10% |
| Proposed Method: Heat | 21.4 | 0.075 | 66% |
| Proposed Method: Dust | 32.5 | N/A | 100% |

Although the airflow required to mitigate the dust can be reduced, the addition of water to the local environment may actually cause the wet-bulb temperature of the area to increase. In any case, the importance of heat as a determinant in the airflow requirements for diesel equipment should not be underestimated.

Additional information regarding diesel equipment and the total airflow requirements may be found in the "Diesel Emissions and Control" course attached to this series of ventilation modules.

Additional information regarding diesel equipment and the total airflow requirements may be found in the "Diesel Emissions and Control" course in these ventilation modules.

As the technological components of battery-powered (electric) heavy mining equipment improve, this equipment may offer an advantage over the more traditionally accepted diesel equipment fleets, particularly in areas where the heat load of equipment poses a risk to miners. For equivalently sized machines, electric vehicles produce approximately 33% of the heat produced by diesel equipment. This disparity may make the difference in economic feasibility for projects that are marginal based on a limited ventilation capacity because of limits on ventilation infrastructure, which may be due to ground conditions, heat load, etc.

**Case Study: Onaping Deep Mine**

Onaping Mine, located in the province of Ontario, Canada lies in the famed nickel rim surrounding the city of Sudbury. Although the climate surrounding the mine is quite temperate (often requiring mine air heating during the winter months, heat will become the driver for ventilation system demand as the mine progresses deeper in the near future. Recognizing this, the mine has investigated alternatives to its current diesel equipment fleet, which represents the single greatest heat source within the mine, and the only significant source that can be easily reduced without greatly impacting mine operations.

Owing to the significant gains in efficiency that can be realized through the use of electric equipment (battery) over their diesel equivalents, the mine performed extensive testing with a battery-powered loader with the goal of establishing the feasibility of implementing a battery-powered fleet in the planned Onaping Deep mine expansion.

Based on the study performed by the mine engineering staff, significant cost savings were shown to be achievable based upon; lower ventilation rates, smaller/fewer raises and drifts, less heat from auto-compression, reduced heat from the mine equipment fleet, and reduced air cooling/refrigeration requirements.

The study shows that for the mine expansion project, a potential capital cost savings of $15M could be realized, with a further $8M per year in realized operating cost savings could be achieved by replacing the diesel equipment fleet with battery-powered electric equivalents.

While changes to battery and equipment technology are advancing rapidly, there are still many challenges that exist for mines that wish to convert their equipment fleets. Challenges with battery-powered mine equipment include; up-ramp hauling, battery charging/change-out stations, cycle times, flexibility and infrastructure requirements and availability.

Aside from choices regarding the type (diesel, electric) and size (power) of mobile equipment underground, the choice of ore and equipment transport also have a significant impact on the ventilation system demand. We have already considered the impacts of a large diesel truck fleet utilized for mineral transport. Conveyor belts also have unique ventilation requirements that must be considered. Large electric belt-drives are important sources of heat, while dust may be generated along the length of a conveyor and especially at transfer points. Of course, many mines will have multiple types of transport systems operating concurrently (e.g., shafts, conveyors, diesel equipment). It is up to the ventilation system engineer to identity the unique requirements for these systems (both individually and in concert)

# 4.4 Fixed Facilities

The fixed facilities, or mine plant, when located underground can also add to the supply of fresh air that is required for a mine. Installations like workshops, offices, crusher stations, fuel bays, warehouses, transformers, etc. all have requirements for ventilation that cannot be ignored or discounted when designing a ventilation system. For some such facilities, (e.g., powder magazines, fuel bays, battery charging stations, etc.) air that is utilized for their ventilation should not be re-used.

Sometimes, requirements for the ventilation of these installations is governed by legislation or corporate standards. Others are set by industry or empirically by the experience of the design engineer.

The following table gives some suggestions for the airflow supplied to common fixed facilities found in the mining environment.

| **Location** | **Minutes Per Air Change** |
|---|---|
| Mine office | 5 |
| Training Room | 6 |
| Warehouse | 7 |
| Electrical Room | 6 |
| Mechanics Shop | 3 |
| Bathrooms | 5 |
| Lunchroom | 5 |

# 4.5 Auxiliary Ventilation

Even in the most well-designed ventilation systems and with the best infrastructure and component installations, auxiliary ventilation will be necessary to provide adequate quantities of fresh air to active development areas and dead-end workings.

The design of auxiliary ventilation systems begins the determination of the total airflow required for the heading. In most cases, this involves separate calculations for each phase of the mining cycle that will occur in any given area (e.g., drilling, loading, mucking). The maximum airflow required (or "worst-case scenario" should be selected for planning purposes and the auxiliary fan and duct sized appropriately.

The selection of a configuration for the system (blowing or exhausting) is determined by the specific conditions or needs of the application and the contaminant that is driving the airflow requirement (i.e., noxious gases, heat or particulates).

Exhausting systems of auxiliary ventilation are particularly well suited to removing particulates from the ambient environment underground. Dust generated at the mining face is pulled directly into the duct without spreading back over the mine workers in the drift, and may be collected in a filter or scrubber installed in-line with the fan, which then discharges clean air back into the circuit.

Because exhausting duct systems operate under negative pressure, it is necessary to use rigid or reinforced ducting, and in some cases where operating pressures are high, the maximum diameter of such ducts may be limited.

Blowing auxiliary ventilation systems are the most commonly installed systems in metal and non-metal mines. They can be utilized with relatively inexpensive and easy to install, flexible ducting. Blowing systems provide clean, fresh air directly to the mine face with a greater velocity than exhausting systems (due to the jet effect when leaving the duct), which aids in cooling and mixing of any gases present.

One of the disadvantages of blowing systems is that they force any contaminants generated at the face to travel through the drift (and over any personnel present) before they are removed from the heading. This can be particularly dangerous in cases where toxic dust or radon are present.

Regardless of the type of system chosen, the installation of the ducting and its components (e.g., inlet bells, elbows, valves and tees) will have a profound impact upon its operation and efficiency. Ducting should be sized correctly (diameter) and installed such that unnecessary bends and restrictions are eliminated.

The duct should always be installed with one end as close to the face as reasonably achievable, within a minimum of 20 m. The other end of the duct should extend approximately 10 m past the intersection with the primary ventilation circuit. No more than 50% of the airflow passing through a drift in the primary circuit should be picked up by the auxiliary ventilation system that is being fed. These measures will prevent air from recirculating through the duct and the potential build-up of contaminants. (DeSouza, et al, 2011).

# 4.6 Blast Fume Clearance

In mines that employ blasting as a component of the mining cycle, the clearance of noxious fumes from production and development headings after blasting is a critical component of ventilation system design. Minimizing the downtime of the mechanized equipment is critical, which must be balanced with the need to protect the workforce from hazardous conditions at all times.

The methods for determining the safe re-entry time after a blast vary greatly, from complex computer simulations performed by specialized software programs, to empirical calculations based on assumptions and experience.

One commonly used method for calculating how much time is required before entering an area after blasting has been originated in South Africa. Peak contaminant levels for CO (5,000 ppm) and NO2 (300 ppm) after blasting were developed empirically and compared against their occupational exposure (or threshold limit) values, 30 ppm and 3 ppm, respectively. Assuming that contaminant levels fall by approximately half for each air change completed, it will take approximately eight air changes in order to dilute the blasting fumes from their peak values to below the level of their respective OEL/TLVs.

**Sample Problem - Blast Fume Clearance**

The Mine Ventilation Society of South Africa (MVSSA) notes that although a value of eight air changes prior to re-entry after blasting is in fact incorporated into South African law, modern explosives produce lower concentrations of gases, meaning that fewer air changes may be required in other locations and suggest that three to five air changes may be sufficient in most cases.

Of course, the safest means for verifying whether an area is safe to enter after blasting is to have a qualified person take readings of the air quality with a calibrated, multi-gas sensor prior to returning to work.

# 4.7 Ventilation Controls

Ventilation controls refer to a list of passive and active devices that determine the direction or distribution of airflow underground. Since airflow follows the path of least resistance in accordance with the basic laws for fluid flow, any air requirements outside of the "natural" flow must be accomplished via these ventilation controls.

The most obvious of these ventilation controls are fans, as primary, secondary and auxiliary installations. Fans supply a pressure differential to a ventilation circuit that causes air to flow from areas of high (relative) pressure to low pressure.

Passive ventilation controls consist of resistances that are added to the ventilation circuit for the purpose of restricting airflow, thereby forcing it through another path. The magnitude of the resistance determines how much flow can pass through the control (see. Square Law). Thus, a well-constructed CMU or shotcrete wall represents an almost impermeable barrier, while a hanging curtain may allow considerable airflow through.

*Regulators* are ventilation controls with an adjustable resistance. These controls are useful in establishing the correct airflow distribution through ventilation circuits with changing airflow requirements throughout the life of a mine or project. Examples of regulators include sliding-type, louver-type and valve-type (butterfly or guillotine). Regulators may be automated for use in VOD systems.

The following Figures show some examples of regulators installed in underground mines.


**Figure 4.10: Drop-board regulator in an underground Uranium Mine.**

**Figure 4.11: Louver regulator in an underground Copper Mine.**

In cases where ventilation flow needs to be restricted, but vehicle access is still needed, a ventilation door is installed. There are many different types of ventilation doors, made from a variety of materials. Doors may be single-swing, double-swing, or roll-up type. They can be constructed of steel, wood, plastic or flexible rubber. Doors can be automated or manually operated, and actuated manually, pneumatically hydraulically or with electric motors.

The following figures show some example of mine ventilation doors.

**Figure 4.12: Underground Ventilation Control Door.**


**Figure 4.13: Underground Ventilation Control Door with mandoor and window.**

It is often desirable to allow vehicle access through a drift that is restricted from flow without allowing flow through the drift when the door is opened (e.g., when the pressure differential is high, or the direction of leakage is from a contaminated air stream to a fresh air source). In these situations, a set of *Airlock* Doors is installed. An Airlock is created by installing two doors in series within an airstream that are separated by a distance at least equivalent to the longest vehicle that will pass through the drift (in higher traffic areas it may be advisable to size the airlock large enough to accommodate several vehicles at once. Doors are electronically interlocked, such that only one door may open at any given time. A vehicle, or vehicles may open the first door, pass in between the doors, then once the first door is closed behind, may open the second door and pass through. In this way, vehicle access can be accomplished without allowing the passage of air between the two sides of the airlock.

Although not commonly found in metal mines, a ventilation *Cross-over* is sometimes useful in separating the air streams in a four-way intersection into two, separate and distinct pathways. Usually, this eventuality is accounted for in the planning/development phase, and when required the two crossing drifts are simply separated by leaving a vertical offset between the excavations. In those rare cases where it becomes necessary to separate the two crossing streams of airflow at an existing intersection, a ventilation Cross-over is constructed by slashing the roof or top of one of the drifts, ramping upwards a distance of two hydraulic diameters from the intersection and then ramping back down again afterwards after a similar distance. A false bottom or “floor” is then constructed at the old roofline of entries, and walls built on either side of the intersection. Along the bottom entry, passage through the drift is the same as before, and the drift profile (cross-section) remains almost the same as before. In the Cross-over drift, the air displaces approximately one drift in height before passing over the original entry and back down again on the other side. If vehicle access is required in this entry, it will be necessary to ramp up the floor on either side of the intersection with material (e.g., gravel, waste rock, concrete, etc.) and heavily reinforce the “floor” of the Cross-over sufficiently to support the weight of the heaviest piece of equipment that will be passing through.

The following figures show how a ventilation Cross-over is constructed in a metal non-metal mine.

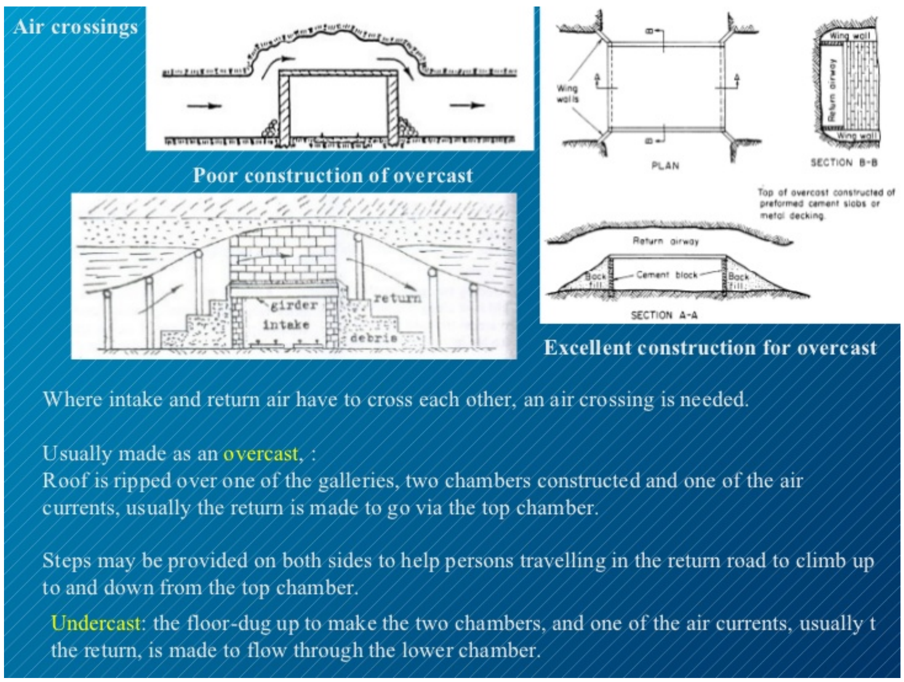



**Figure 4.14: Overcast Construction for the separation of air paths at a common intersection.**

# 4.8 Fan Selection

The selection of fans for use in the ventilation system, whether they be for use as primary, booster (secondary) or auxiliary ventilation will have a profound impact on many aspects of the operation. Generally, flow quantity is the deciding factor in fan selection, but the operating pressure(s), the location (surface, underground, proximity to raises, bends, etc.), the noise profile and the overall robustness of the installation will have an effect on the suitability of any given fan for any given application.

Consider a new primary fan installation that will be installed on the surface above an existing mine property. Although the pressure is relatively low, and an axial fan provides the highest possible efficiency, a centrifugal fan was chosen. The installation being close to a town, and it not being possible to install the fan underground, the lower noise profile and vertical discharge of the centrifugal fan were enough to make it the choice for this particular situation. This is just one example of how fan attributes can contribute to making a selection for a unique installation, and how each installation must be considered individually.

Additional, detailed information on fans and fan properties can be found in the “Fans” course attached to this series of educational modules.

The following figures show some examples of fan installations underground.


**Figure 4.15: Underground booster fan for production panel with pressure-relief door.**


**Figure 4.16: Underground booster fan installation (inlet side).**

**Figure 4.17: Underground booster fan installation showing backflow dampers (outlet side).**

# 5.0 Wrap-up and Quiz

The following quiz will test your proficiency in meeting the learning objectives.  Quiz questions are taken from the module content.

**MNM Ventilation System Design (https://canvas.instructure.com/courses/1049599/quizzes/1866835)**

# Ventilation Economics

[Jump to Today]  Edit

The following course provides an overview of Ventilation Economics and the basis for evaluating ventilation projects on an economic cost/benefit basis. Fundamental principles of economic theory, including Net Present Value and Discounted Cash Flow are discussed. This course is part of a larger suite of courses covering a variety of topics in mine ventilation.

### Learning Objectives

1. **Explain the time value of money**
2. **Create and utilize cash flow diagrams**
3. **Articulate feasibility analysis and equivalence calculations**
4. **Describe risk analysis and alternative analysis**
5. **Perform basic optimization of a project**

**Get Started**

**About Aeolus**

# Course Summary:

| Date | Details |
|---|---|
| | Ventilation Economics Quiz (https://canvas.instructure.com/courses/1199718/assignments/6912780) |

# 1.0 Introduction

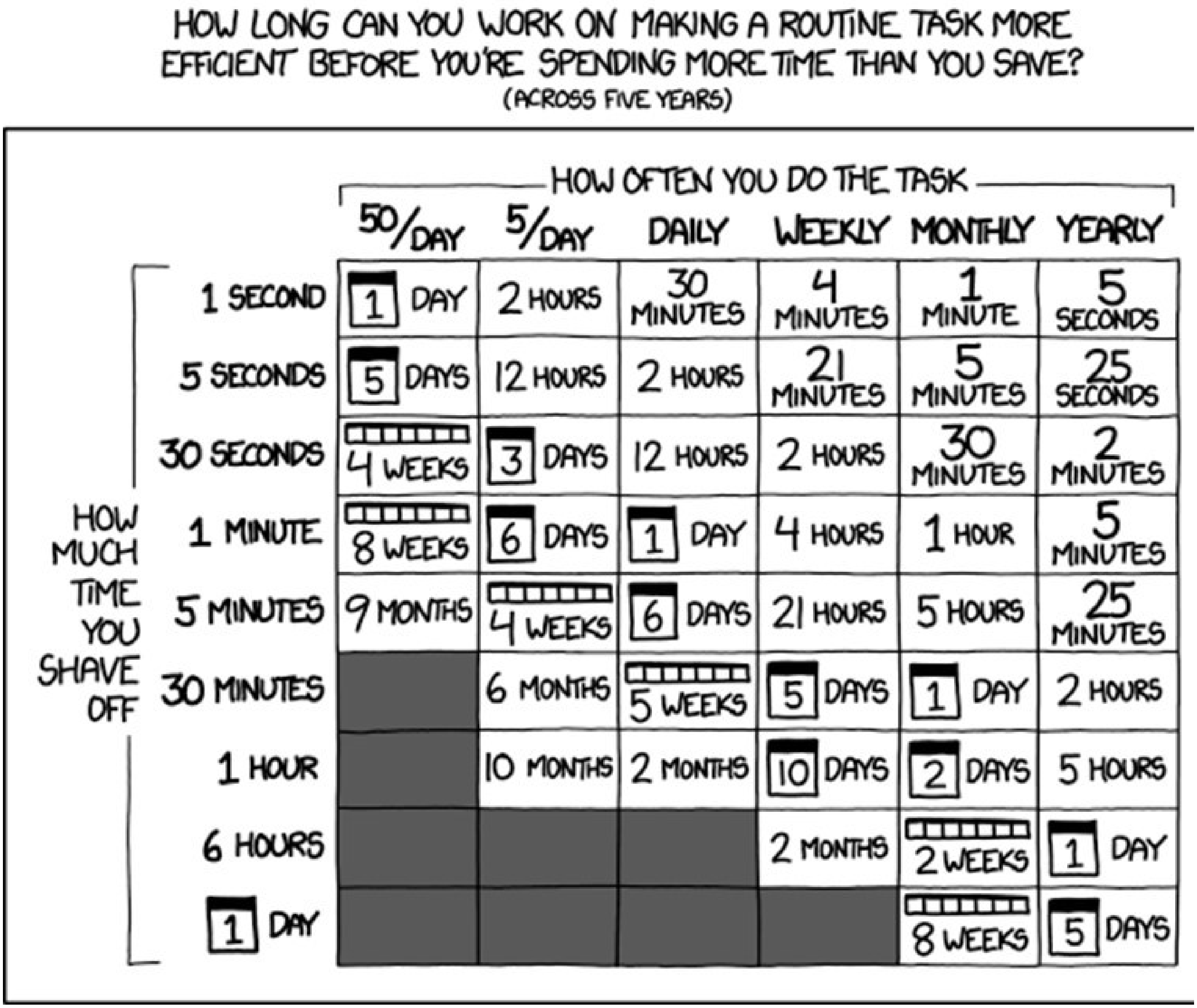
Economics is the study of trade-offs; and of finding the best value at the time under the existing (or planned) conditions. Some of the parameters needed to understand the cost-benefit analysis include; Constraints, Time Value of Money, Cash flow, Payback period, Risk and Optimization Measures.

Over time money is able to purchase more or less of a certain commodity or “goods”. Money has value only because it can be exchanged for services or for goods. The exchange can take place now or in the future, which has an impact on the amount of goods or services that can be obtained at the time of the exchange. The **Time Value of Money** is expressed as an interest rate and describes either inflation (money is worth less) or deflation (money is worth more).

# 1.1 Optimization Methodology

While it is impossible to find the optimum cost to price point for a mining operation's life, it is possible to develop the optimization methodology for the mine site and the mine method that should be revisited regularly.

Figure 1.1 shows a visual representation of the benefits of making a process or task more efficient versus the time spent making those gains (optimizing).




| | 50/DAY | 5/DAY | DAILY | WEEKLY | MONTHLY | YEARLY |
|---|---|---|---|---|---|---|
| 1 SECOND | 1 DAY | 2 HOURS | 30 MINUTES | 4 MINUTES | 1 MINUTE | 5 SECONDS |
| 5 SECONDS | 5 DAYS | 12 HOURS | 2 HOURS | 21 MINUTES | 5 MINUTES | 25 SECONDS |
| 30 SECONDS | 4 WEEKS | 3 DAYS | 12 HOURS | 2 HOURS | 30 MINUTES | 2 MINUTES |
| 1 MINUTE | 8 WEEKS | 6 DAYS | 1 DAY | 4 HOURS | 1 HOUR | 5 MINUTES |
| 5 MINUTES | 9 MONTHS | 4 WEEKS | 6 DAYS | 21 HOURS | 5 HOURS | 25 MINUTES |
| 30 MINUTES | | 6 MONTHS | 5 WEEKS | 5 DAYS | 1 DAY | 2 HOURS |
| 1 HOUR | | 10 MONTHS | 2 MONTHS | 10 DAYS | 2 DAYS | 5 HOURS |
| 6 HOURS | | | | 2 MONTHS | 2 WEEKS | 1 DAY |
| 1 DAY | | | | | 8 WEEKS | 5 DAYS |

**Figure 1.1: Schematic of efficiency gains based on task frequency and time spent optimizing.**

# 1.2 Identifying Constraints

There are a wide variety of constraints on ventilation projects, any and all of them must be enumerated and where possible quantified during an economic analysis.  Some examples of constraints include:

- Air quantity delivered and measurable to the working area(s)
- Air quantity delivered per running diesel engine
- Spacing necessary between intake and exhaust
- Diesel cost and variability
- Electricity costs, daily and seasonal variability
- Tunnel shape and size
- Ventilation controls already in place
- Cost ($)

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- Diesel cost and variability
- Electricity costs, daily and seasonal variability
- Tunnel shape and size
- Ventilation controls already in place
- Cost ($)

# 1.3 Bottleneck Analysis

A common optimization technique is to identify bottlenecks in the system.  For ventilation systems, this is often finding branches or districts with exceptionally high resistance or leakage values.  Fan settings are often adjusted as a way to increase pressure delivered to a mine or area, which can mask the underlying leakage or resistance.  A Bottleneck Analysis is never completed, when the worst problem in the system is identified and remedied- what was previously the second-worst problem in the system is now the top problem.  Sometimes, a new problem has been introduced.

# 1.4 Risk Analysis

Optimization with direct respect to risk is an important technique.  In this analysis procedure, constraints are further analyzed with respect to the likelihood that the defined limits will be achieved.  Safety factors on ventilation controls are added such that the likelihood of an undesired event is minimized.  Building excess capacity into the system is also a risk to the operation because cost is still a constraint.

**Next Module:**
2.0 Project Engineering

# 2.0 Introduction

A Project Engineer is a manager with engineering qualifications. They are often required to be registered as a Professional Engineer or a Competent Person by the relevant regulatory authorities. Project Engineers are responsible for:

*"…execution of one or more simultaneous projects in accordance with a valid, executed contract, per company policies and procedures and work instructions for customized and standardized tasks."*

The duties of a Project Engineers duties may include schedule preparation and resource forecasting for engineering activities; assurance of financial forecast accuracy, which tie to project schedules; management of team resources (employees, contractors, vendors and other project personnel); and assurance that projects are completed according to project plans.

# 2.1 Feasibility Analysis

Project Engineering often involves the evaluation of the **feasibility** of a given project, activity or undertaking. Feasibility is a measure of a project's viability, practicality or achievability. In order to determine the feasibility of a project, a Feasibility Analysis is conducted. The feasibility analysis may include Economic Analyses, Financial Analyses and other, Intangible Analyses.

"Engineering economy" is the economic evaluation of investment situations or opportunities. They are concerned with estimated costs and projected profits at each stage of project evaluation. At each point, the question "is there sufficient merit to proceed to next stage?" is asked, and answered before proceeding. It frequently becomes more detailed as the project moves toward becoming a real undertaking.

A project may be considered feasible if the projected earnings are sufficient to justify the investment and operating and maintenance costs. If the return on investment isn't sufficiently high enough to attract investment, or outpace the return on other options for capital expenditures, the project is unlikely to ever by completed (or undertaken).

Since some parameters are unknown and must be estimated, risk is introduced. Examples of risk include a very sensitive commodity price, or poorly defined geologic resource, etc. In these cases, the most common risk analyses utilize statistical tools to better quantify the risk(s) involved.

Another important aspect of Feasibility Studies and analyses, are that they serve as a point of comparison by which proposed activity and alternative investments and options are measured.

# 2.2 Financial Analysis

Financial Analyses are concerned with activities required to secure financial backing for project. These analyses answer the questions of how, where and at what cost the investment funds are obtained? For large mining operations, this often requires very substantial financial investment. The large financial commitment for mining projects effectively limits the number and size of financial organizations that may be involved. This explains why large, multinational banks and companies are so often involved in mining finance.

# 2.3 Intangible Analysis

Intangible Analyses are those concerned with "tough-to-define" factors that can nonetheless impact mine success. These types of factors include:

- What stakeholders are involved and how are they impacted?
- They can be very important, but difficult to quantify
    - environmental, political, preferences, urgency, goodwill, health/safety, prestige, etc.

Some miners view this as an added cost to development, operation and closure; however, not everything measured in dollars. Some intangible project factors may prove to be insurmountable, and ultimately lead to the failure, or abandonment of the project.

# 3.0 Cash Flow Diagrams

Mining engineers are often called upon to perform an "economic analysis" of a particular project or activity. The elements of an economic analysis include:

- Revenue
  - Includes any earnings associated with the proposed project.
- Expenses
  - Includes capital investments and Operation and Maintenance (O & M) costs associated with the proposed project.

A "cash flow diagram" (CFD) is a tool used by accountants and engineers to represent the transactions which will take place over the course of a given project. These diagrams graphically illustrate the time sequence of expected receipts and disbursements. Transactions can include initial investments, maintenance costs, projected earnings or savings resulting from the project, as well as salvage and resale value of equipment at the end of the project.

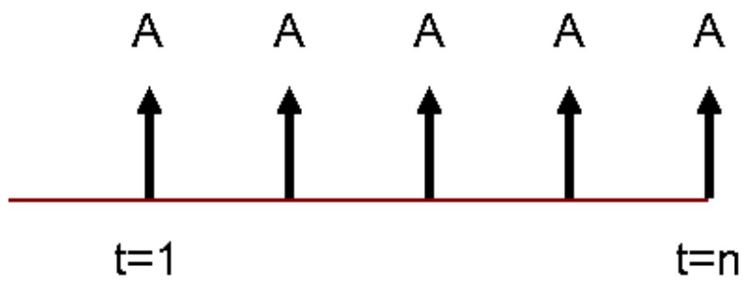
The following conventions are used when constructing a cash flow diagram:

- The horizontal axis (Time) is marked off in equal increments up to the duration of the project.
- Two or more transfers in the same period are combined and placed end to end.
- Expenses before t=0 ("sunk" costs) may not be included unless they have tax consequences.
- Receipts are upward arrows, while disbursements are downward arrows.
- Arrow length is proportional to magnitude of the cash flow.

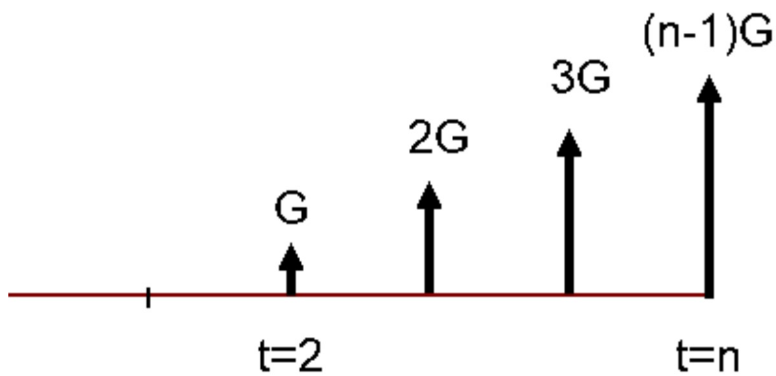
It is customary to assume that all receipts and disbursements occur at the end of the year. This is known as the "**Year End Convention**". Exceptions to this rule include cash flows specifically occurring at time = 0 (e.g., purchase cost). A more detailed analysis has been shown to improve the precision only slightly, and does not justify the additional work necessary.

- Single
  - One payment at any time along timeline.

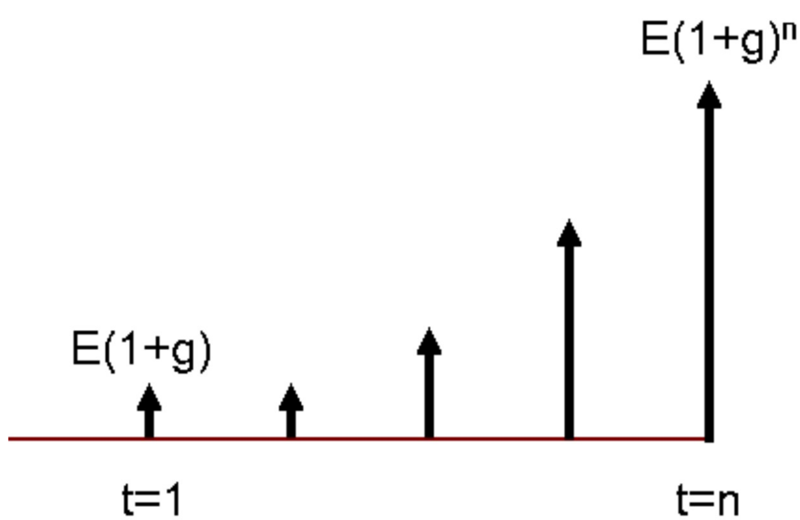

- Uniform
  - Equal transactions (A) starting at t=1 and ending at t=n.




- Gradient
  - Transaction (G) starting at t=2 and building by same amount.




- Exponential
  - Transaction (E) starting at t=1 and changing exponentially.




Assumptions:

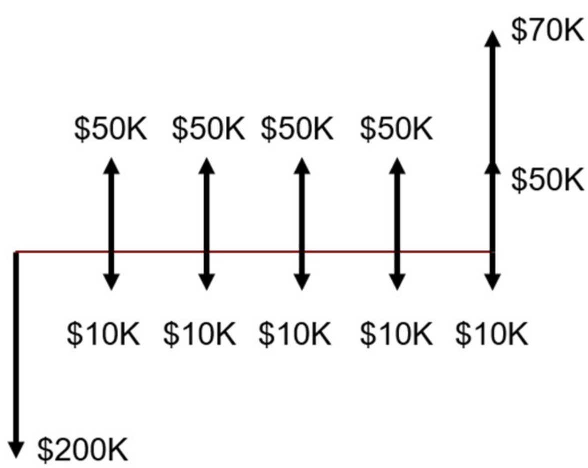
- Year-end convention is applicable.
- No inflation exists now or any time in project.
- Effective interest rate will be constant.
- Non-quantifiable parameters are ignored.
- Investment funds are available.

- Excess funds continue to earn interest at the effective rate.
- These assumptions, which may or may not be wholly valid, are made regardless for the benefit of finding a solution.

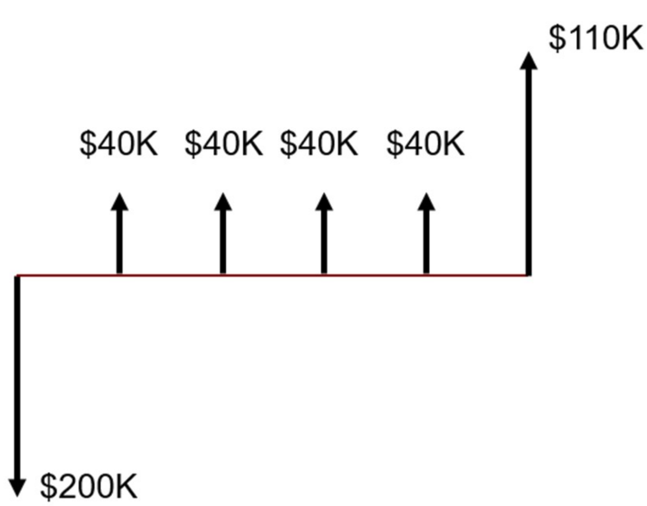
Cash flow diagrams may also be used to represent payment schedules for bonds, mortgages and other types of loans. They are often used with a “break even sheet” and/or a “balance sheet” to see where money is being made or lost. CFDs can also be used to evaluate results of different financial scenarios, for instance, to see if changing variables are successful or not in increasing future profits.

**Example:**

A new resource has been identified that requires the installation of a new fan. A new fan will cost $200,000 when purchased. The maintenance cost is expected to be $10,000 per year. The revenue realized from the additional airflow is expected to be $50,000 for five years, after which the fan will have a salvage value of $70,000. What does the CFD look like?




OR



# 4.0 Time Value of Money

Cash flow analysis must reflect the “time value” of money. This is because today’s dollars are not worth the same as future, or past dollars. Whenever money is used over time (e.g., investments, loans, etc.), it is necessary to consider the effects of compound interest. The “value” of money at different dates or points in time must be established; failure to do so will result in poor economic decisions.

# 4.1 Equivalence

Equivalence is heavily dependent on the interest or discount rate employed. The effective interest rate is the interest rate used in the actual calculations. This is distinguished from other rates (e.g., nominal rate) not used in calculations. It may be compounded yearly (most common), monthly, weekly, etc.

# 4.2 Discounted Cash Flow

Discounting is essentially the same as present worth in an economic evaluation. A "discounted cash flow" is the net inflow or outflow of money during a given period that considers the time value of money. When comparing different projects or project alternatives, the discounted cash flow should be used to avoid poor decisions. Comparisons should always be made on the basis of "identical" dollars; that is, dollars with equivalent worth. It should be obvious that a mathematical approach is needed, so that projects with revenue and expenses occurring at different times can be compared.

For example, if the annual discount rate is 5%, which option is preferred?

- $100 now
- $105 paid in one year
- $110 paid in two years

As it turns out, these options are "equivalent". Procedure for determining equivalent amount is known as "discounting".

# 5.0 Equivalence Calculations

Equivalence calculations have their own “language” of terms and terminology, and an understanding of this language is important in order to perform and evaluate these calculations.

# 5.1 Important Terminology/Symbols

The following comprise some of the most commonly used symbols and terms encountered in Equivalence Calculations:

- i = period compound interest rate
- t = time duration
- P = present (principal) sum of money (usually at t=0)
- F = single sum of money at designated future date
- A = amount of each payment in a uniform series of equal payments at the end of each period
- n = number of interest compounding periods (may represent annual, semiannual, quarterly, or even daily periods)

# 5.2 Examples of Equivalence

Future value (F) of a present sum of money (P) subject to an interest rate (i) is:

- Year 1: $F = P(1+i)$
- Year 2: $F = P(1+i)(1+i) = P(1+i)^2$
- Year 3: $F = P(1+i)(1+i)(1+i) = P(1+i)^3$
- Year n: $F = P(1+i)^n$
- Conversely, present value of future sum of money realized after n periods of time is:
- $P = F/(1+i)^n$

  $(1+i)^n$ = single payment compound amount

  $1/(1+i)^n$ = single payment present worth

These F and P equations can the thought of as conversion factors:

- compound amount factor = $(1+i)^n$
  - converts present into future
- present worth factor = $1/(1+i)^n$
  - converts future into present

For example, how much must be put into a 10% savings account today to have $10,000 in 5 years? Or, in other words, what is the equivalent present worth of $10,000 five years from now if money is worth 10%.

$P= F/(1+i)^n =\$10,000/(1+0.1)^5 = \$6,209$

Cash flows that occur yearly are common in economic analyses. A cash flow (A) that repeats yearly is known as an “annual amount.” For equipment, these may represent “operation and maintenance” or “O&M” costs. For a business, these may be “general, selling and administrative” or “GS&A” expenses. While these amounts could be calculated and summed as before, it is often easier to use one of the “uniform series factors”.

For example, if an amount (A) is invested at year end at an interest rate (i), then:

- Year 1: $F = A$
- Year 2: $F = A + A(1+i)$
- Year 3: $F = A + A(1+i) + A(1+i)^2$
- Year n: $F = A[(1+i)^n-1]/i$

A “sinking” fund set up to provide a target amount after n series of payments is:

$A = F [i/((1+i)^n-1)]$

$[(1+i)^n-1]/i$ = uniform series compound amount

$i/[(1+i)^n-1]$ = uniform series sinking fund deposit

- Similarly, to determine the uniform year end payment (A) realized for n periods from a single investment (P), we find:

$A = P(1+i)^n\ [i/((1+i)^n-1)]$

$= P[i(1+i)^n)]/[(1+i)^n-1]$

Likewise, a single present sum (P) equal to a uniform series of n year-end payments (A):

$P = A\ [(1+i)^n-1)]/[i(1+i)^n]$

$[i(1+i)^n)]/[(1+i)^n-1]$ = capital recovery

$[(1+i)^n-1)]/[i(1+i)^n]$ = uniform series present worth

Question:

If yearly O&M costs are $25k, what is the present worth of these costs over a 12-year period if the interest rate is 8%?

Solution:

$P = A\ [(1+i)^n-1]/[i(1+i)^n]$

$= \$25{,}000\ [(1+0.08)^{12}-1]/[0.08(1+0.08)^{12}]$

$= \$25{,}000 \times 7.5361 = \$188{,}400$ (negative)

Calculations involving annual values:

- uniform series compound amount

$= [(1+i)^n-1]/i$

- converts annual into future

- uniform series sinking fund

$= i/[(1+i)^n-1]$

- converts future into annual

- uniform series present worth

$= [(1+i)^n-1]/[i(1+i)^n]$

- converts annual into present

- uniform series capital recovery

$=[i(1+i)^n]/[(1+i)^n-1]$

- converts present into annual

Another common complication involves uniformly increasing cash flow. For example, annual maintenance is $10,000 and will increase by $1,000 each year. These types of calculations are performed using a “uniform gradient factor.”

- uniform gradient present worth
- converts gradient to present

$$\frac{(1+i)^n-1}{i^2(1+i)^n}-\frac{n}{i(1+i)^n}$$

- uniform gradient future worth
- converts gradient to future

$$\frac{(1+i)^n-1}{i^2}-\frac{n}{i}$$

- uniform gradient uniform series
- converts gradient to annual

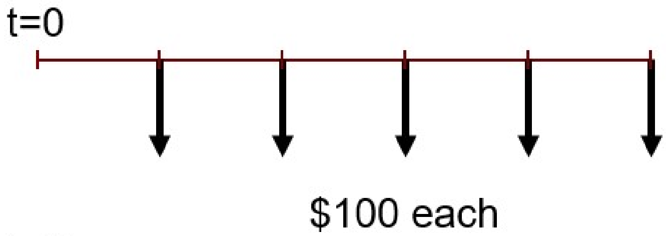
$$\frac{1}{i}-\frac{n}{(1+i)^n-1}$$

Question:

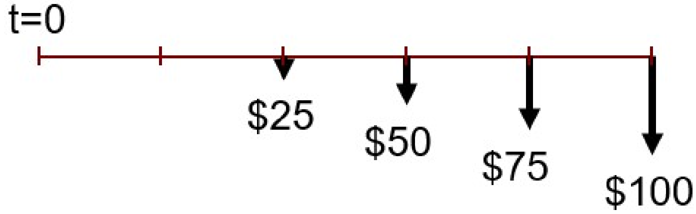
Maintenance on an old regulator is $100, but will increase $25 each year. At 10% interest, what is the present worth of 5 years maintenance?

Solution:

Problem must be solved in two parts:




$$P = A\left[\frac{(1+i)^n - 1}{i^2(1+i)^n}\right] = 100(\ldots$$




$$P = G\left[\frac{(1+i)^n - 1}{i^2(1+i)^n} - \frac{n}{i(1+i)\ldots}\right.$$

Present Worth = $379 + $172 = $551 (negative)

# 6.0 Standard Tables / Notation

Economic calculations can be simplified using standard tables and notation. Rather than actually writing formulas, it is common to substitute functional notation. For example, the compound amount factor can be symbolically written as:

$F = P(F/P, i\%, n)$

Similarly, the present worth factor can be symbolically written as:

$P = F(P/F, i\%, n)$

Values of these cash flow (discounting) factors are tabulated in standard tables.

Table 6.1 gives an example of a standard table for discounting calculations.

**Table 6.1: Standard Table.**

| **Factor Name** | **Converts** | **Symbol** | **Formula** |
|---|---|---|---|
| Single Payment Compound Amount | P to F | (F/P, i%, n) | $(1+i)^n$ |
| Single Payment Present Worth | F to P | (P/F, i%, n) | $\frac{1}{(1+i)^n}$ |
| Uniform Series Sinking Fund | F to A | (A/F, i%, n) | $\frac{i}{(1+i)^n - 1}$ |
| Capital Recovery | P to A | (A/P, i%, n) | $\frac{i(1+i)^n}{(1+i)^n - 1}$ |
| Uniform Series Compound Amount | A to F | (F/A, i%, n) | $\frac{(1+i)^n - 1}{i}$ |
| Uniform Series Present Worth | A to P | (P/A, i%, n) | $\frac{(1+i)^n - 1}{i(1+i)^n}$ |
| Uniform Gradient Present Worth | G to P | (P/G, i%, n) | $\frac{(1+i)^n - 1}{i^2(1+i)^n} - \frac{n}{i(1+i)^n}$ |
| Uniform Gradient Future Worth | G to F | (F/G, i%, n) | $\frac{(1+i)^n - 1}{i^2} - \frac{n}{i}$ |
| Uniform Gradient Uniform Series | G to A | (A/G, i%, n) | $\frac{1}{i} - \frac{n}{(1+i)^n - 1}$ |

To some, these standard tables can be confusing. The best way to remember the notations is to treat them "algebraically."

Think of this: F = P(F/P, i%, n)

…like this: F = P x F/P

This concept also works for calculating other combinations, i.e.: A = F (A/F, i%, n)

…A = F (P/F, i%, n) x (A/P, i%, n)

Question:

What factor will convert a future cash flow ending at t=8 to a present value at t=0?

Assume the effective annual interest rate is 10%.

Answer:

From the formula chart, we find:

$$(P/F, i\%, n) = \frac{1}{(1+i)^n} = \frac{1}{(1+0.1)^8} = 0.4665$$

Alternatively, from the tabular values,

(P/F,10%,8) = 0.4665.

Question:

If yearly O&M costs for the main mine fan are $2,500, what is the present worth of these costs over a 12 year period if the interest rate is 8%?

Solution:

P = A [P/A, 8%,12]

= $2,500 x 7.5361 = $18,840 (negative)

# 6.1 Reporting

When reporting values derived from NPV calculations, fractional parts of a dollar (cents) are often omitted from economic calculations. Values are generally reported to four significant figures (unless first digit is 1, in which case five are used).

- $49 not $49.43
- $93,450 not $93,454
- $1,289,700 not $1,289,673

# 7.0 Non-Standard CFDs

The established convention for CFDs holds for many common scenarios, including:

- Single Payment
- Uniform Series
- Gradient

Unfortunately, not all situations fit the standard cash flow expressions.  Under these “Non-Standard cash-flow scenarios, multiple options exist, including:

- Convert individual receipts and disbursements back/forward using standard formula
- Derive new algebraic formula using mathematical relationships

Some examples of non-standard cash flow diagrams include:

- Stepped
- Missing/Extra
- Delayed/Premature
- Year Beginning

# 7.1 Stepped Cash Flow

Stepped cash flows make a step change at some future point in time. This can be handled by the "superposition" of cash flow diagrams.

Example

Investment = \$1000 at t=0

Return = \$100 for t=1 to 5

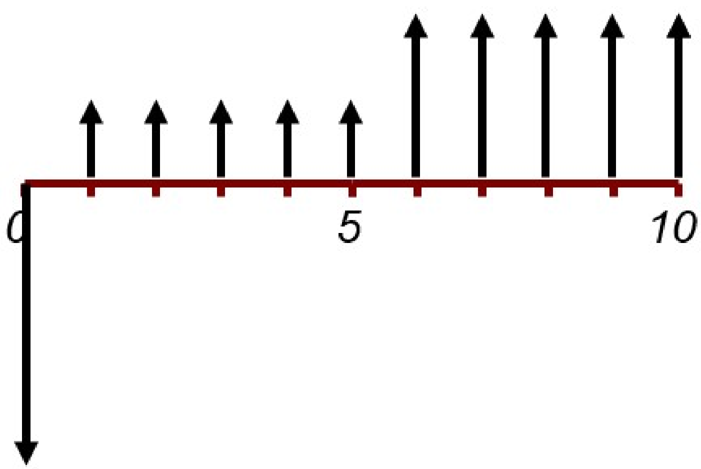
\$200 for t=6 to 10

Superposition gives:

\$200 for t=1 to 10

-\$100 for t=1 to 5

P = -1000 + 200(P/A,i%,10)-100(P/A,i%,5)



# 7.2 Missing / Extra Cash Flow

Missing/Extra Cash flow values may appear and disappear in time. These cash-flows can also be handled by "superposition" of cash flow diagrams.

Example

Return = \$200 for t=1 to 10 (except t=9)

Superposition gives:

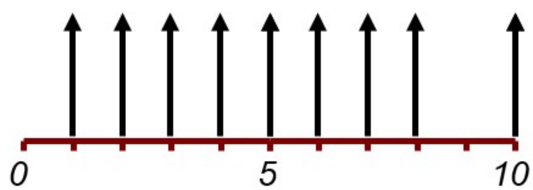
\$200 for t=1 to 10

-\$200 for t=9

P = 200(P/A,i%,10)-200(P/F,i%,9)

or

P = 200(P/A,i%,8)+200(P/F,i%,10)



# 7.3 Delayed / Premature Cash Flow

Delayed/Premature Cash flow matches a standard CFD, but starts (stops) sooner than it should.

Handled using either "superposition" or "projection" (timeline shift) methods.

Example

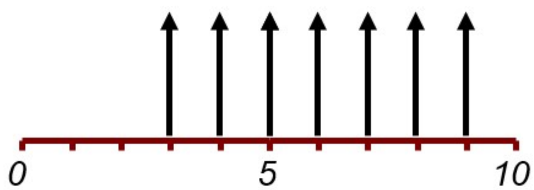
Return = $200 for t=3 to 9

Projection calculates the present value (P') at t=2, which gives:

P' = 200(P/A,i%,7)

P = P'(P/F,i%,2)

= 200(P/A,i%,7)(P/F,i%,2)



# 7.4 Year Beginning Cash Flow

Year Beginning cash-flows of equal amounts starting at t=0 and ending at t=n-1. These are solved via "superposition".

Example

Return = $200 for t=0 to 9

For present worth, superposition gives:

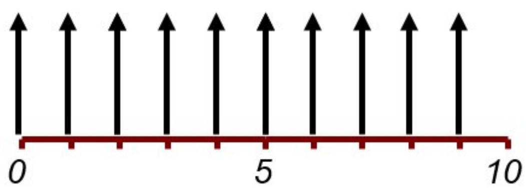
P = $200+$200(P/A,i%,9)

For future worth (n=10), superposition gives:

F = $200(F/P,i%,10)

+ $200(F/A,i%,10)

$200



# 7.5 Exponential Cash Flow

Exponential Gradient cash-flow have payments that grow or decay exponentially.

Consider $100 receipts that increase annually by a fixed percentage of 15%.

Year 1 - $100 (1.15)

Year 2 - $100 (1.15)(1.15)

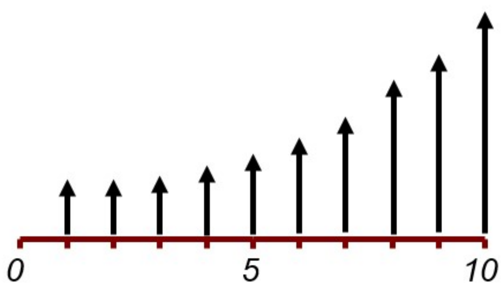
Year 3 - $100 (1.15)(1.15)(1.15)

Year n - $100 $(1+g)^n$

Please note that:

g = exponential growth rate

(This type of cash-flow is rarely seen in economic justification projects assigned to engineers)



# 8.0 Economic Measures

Several different “value” measures are used in engineering project worth analysis. Each measure provides important information on the “attractiveness” of a potential project. Each method has benefits and shortcomings, and the choice reflects corporate or personal philosophy of the entity performing the calculation(s).

Examples of common measures include:

- Net Present Value (NPV)
- Payback Period (PP)
- Present Worth Index (PWI)
- Rate of Return (ROR)

# 8.1 Net Present Value

NPV is the economic value expected to be generated by the project at the time of measurement. It represents the value being added to the company by making the investment. Typically, the decision rule is to only invest in projects where the NPV is positive.

Larger investments normally have a larger NPV (ranking favors large investments). This method of comparison does not consider length of time to achieve the calculated value, and it is highly dependent on interest rate (i.e., cost of capital) used.

Example

- Yr 0: $-\$100/(1+0.1)^0 = -\$100K$
- Yr 1: $(\$32-\$5.6)/(1+0.1)^1=\$24.0K$
- Yr 2: $(\$32-\$5.6)/(1+0.1)^2=\$21.8K$
- Yr 3: $(\$32-\$5.6)/(1+0.1)^3=\$19.8K$
- Yr 4: $(\$32-\$5.6)/(1+0.1)^4=\$18.0K$
- Yr 5: $(\$32-\$5.6)/(1+0.1)^5=\$16.4K$
- Yr 6: $(\$32-\$5.6)/(1+0.1)^6=\$14.9K$

Sum=$15.0K

NPV = $15.0K (NPV>0)

In this case, it may be considered better to make the investment than to do nothing.

# 8.2 Payback Period

This method of comparison measures the time that the net investment will be at risk (simple version uses no interest). The longer the payout period, the greater the chance of some unfavorable circumstance. A three-year payback for ten-year project represents 30% of project life committed to recouping investment and 70% devoted to creating value.

Measures up to time of payback (disregards cash flows received after payout period) and does not directly measure the value created by the project (i.e., is a risk indicator only). It is also highly dependent on interest rate used.

Example

- Yr 0: $-\$100/(1+0.1)^0 = -\$100K$ [-$100K]
- Yr 1: $(\$32-\$5.6)/(1+0.1)^1=\$24.0K$ [-$76.0K]
- Yr 2: $(\$32-\$5.6)/(1+0.1)^2=\$21.8K$ [-$54.2K]
- Yr 3: $(\$32-\$5.6)/(1+0.1)^3=\$19.8K$ [-$34.4K]
- Yr 4: $(\$32-\$5.6)/(1+0.1)^4=\$18.0K$ [-$16.3K]
- Yr 5: $(\$32-\$5.6)/(1+0.1)^5=\$16.4K$ [+$0.1K]
- Yr 6: $(\$32-\$5.6)/(1+0.1)^6=\$14.9K$ [+$15.0K]

PP»5 yrs (PW turns from negative to positive)

A Longer payback denotes higher risk (i.e., have to nearly complete the project to get your money back).

# 8.3 Present Worth Index

PWI is the ratio of present value of cash inflows to the present value of the cash outflows. It measures the relative attractiveness of projects per dollar of investment (i.e., how efficient is each dollar invested in producing value). It is designed to address the limitation of NPV cited previously. PWI is one of the best measures for comparing and deciding between mutually exclusive projects.

However; PWI is not a good indicator of project significance.  It is highly dependent on interest rate (i.e., if wrong rate used, wrong project may be selected).

Example

- Yr 0: \$0 -\$100K
- Yr 1: $\$32/(1+0.1)^1$=\$29.1K -$\$5.6/(1+0.1)^1$=\$5.1K
- Yr 2: $\$32/(1+0.1)^2$=\$26.5K -$\$5.6/(1+0.1)^1$=\$4.6K
- Yr 3: $\$32/(1+0.1)^3$=\$24.0K -$\$5.6/(1+0.1)^1$=\$4.2K
- Yr 4: $\$32/(1+0.1)^4$=\$21.9K -$\$5.6/(1+0.1)^1$=\$3.8K
- Yr 5: $\$32/(1+0.1)^5$=\$19.9K -$\$5.6/(1+0.1)^1$=\$3.5K
- Yr 6: $\$32/(1+0.1)^6$=\$18.1K -$\$5.6/(1+0.1)^1$=\$3.2K

Sum=\$139.4K Sum=\$124.4K

PWI = 139.4/124.4 = 1.12

In this case, it is better to invest than to do nothing, but this is not really a high value project (i.e., PWI just over unity).

# 8.4 Rate of Return

For ROR comparisons, we first must ask the question, “What is the meaning of ‘return?’” If $100 is invested at 5% interest for one year, then:

$F = P(1+i)^n = \$100(1.05) = \$105$

The “return on investment (ROI)” is:

$105 - $100 = $5

However, the worth from a CFD shows:

P = -100 + 105(P/F,5%,1)

= -100 + 105(0.9524) = 0

Thus, the present worth is <u>zero</u> even with a 5% ROI (i.e., ROI may not be a good measure of value).

Let’s look at another example: say you were offered $120 for the use of $100 for one year.

P = -100 + 120(P/F,5%,1)

= -100 + 120(0.9524) = $14.29

The present worth is $14.29. or, to put is another way, the present worth of $15 difference in ROIs. It is the actual earned interest rate is the rate that makes the present worth of the alternative equal to zero. This is called the “rate of return (ROR)”.

P = -100+120(0.8333) = 0

ROR = 20% since (P/F,20%,1)=0.8333

Another way to look at this is to consider that the present worth is the amount needed to dissuade one from making an investment. For example, investing $100 in an alternative that would give $120 in one year would be dissuaded by a payment of $14.29 or more at t=0. Placing the initial investment ($100) and present worth ($14.29) into a bank at the same interest rate (5%) would yield the same ROI ($20).

The selection of an appropriate interest rate is difficult in engineering economics. It is often taken as the average ROR realized by the company in the past. If the rate is not known, a company may specify an “minimum attractive rate of return (MARR)”. MARR is then used as the effective interest rate in economic analyses. Projects with negative present worth are undesirable, while those with positive present worth are desirable. A positive present worth is equivalent to an increase in the average earning power of invested capital.

This represents the discount rate which equates the project's “discounted” net cash inflows with its “discounted” net cash outflows. It measures the efficiency of the project in producing value without reference to any predetermined cost of capital. The decision rule is to have a ROR greater than the MARR for a given project.

ROR analyses favor projects with a quick payout or short-term in nature. It assumes project cash flows are reinvested at same rate of return as the project generates. This method is best when used together with other methods.

Example

- Yr 0: -$100/(1+?)$^0$
- Yr 1: ($32-$5.6)/(1+?)$^1$
- Yr 2: ($32-$5.6)/(1+?)$^2$
- Yr 3: ($32-$5.6)/(1+?)$^3$
- Yr 4: ($32-$5.6)/(1+?)$^4$
- Yr 5: ($32-$5.6)/(1+?)$^5$
- Yr 6: ($32-$5.6)/(1+?)$^6$

*Solve for value of ? that gives NPV=0.*

In this case, it would be better to invest if this value exceeds the minimum attractive rate of return (MARR).

ROR analysis does have some significant limitations. The method does not fully reflect total value of future profits for projects with a long life. This can be significant when the ROR is high and payback is short. Mining projects often fall into this category. The method does not indicate how “quickly” investment funds are recovered. For example, the same ROR can be obtained for 1 or 100-year projects. In these cases, a quick return on investment is safer, since it is possible to predict near-future events much more accurately.

# 8.5 Summary

In general, it is important to use a variety of economic measures (e.g., NPV, ROR, PP, PWI) for project investment analysis. Each measure provides important information on the “attractiveness” of any potential project.

- **NPV** = today’s worth in dollars
- **ROR** = project return on investment
- **PP** = time to recoup initial investment and avoid further risk
- **PWI** = efficiency in generating value per unit of investment for project comparisons

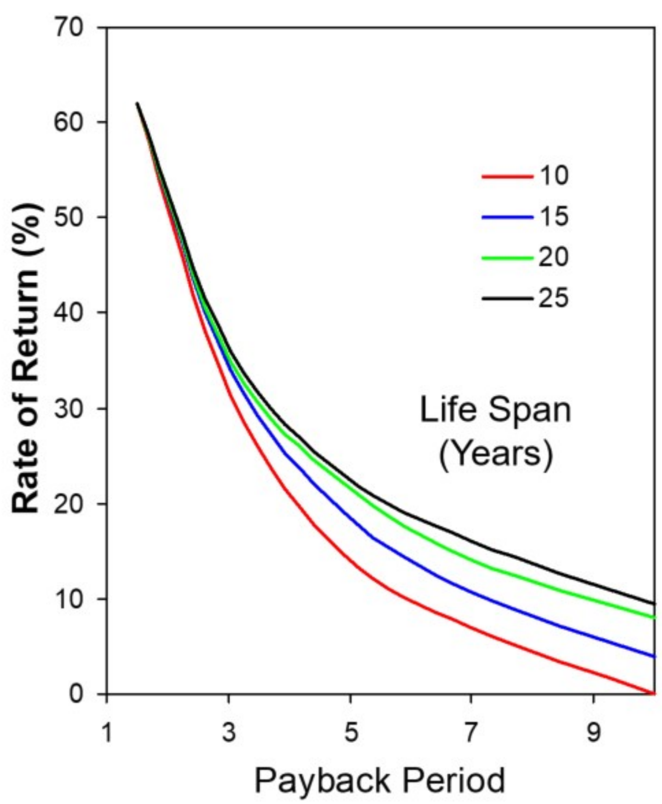
Remember, investors want to:

- maximize rate of return
- minimize risk (get money back quickly)

One way to reduce risk is to have a quick “payback period”. Therefore, at a minimum, it is useful to calculate both the **ROR** and **Payback Period**.

Note that ROR is inversely proportional to payback period. Their curves run together as ROR increases. For the same payback period, a longer project gives a higher ROR. But remember, a longer project life may include more uncertainty and greater risk.

The relationship between ROR and PP is shown on Figure 8.1.



# 9.0 Comparison of Alternatives

An important use of engineering economics is the selection of a "best" investment or project from several alternatives. Consider the following example:

Project A

- Present Cost = $10,000 today
- Future Payback = $11,500 in two years

Project B

- Present Cost = $8,000 today
- Future Payback = $4,500/yr for two years

Which project represents the best value, or investment? In order to answer this question, we must consider whether the alternatives are mutually exclusive, that is, that only one of the options can be selected. Another possible option is to do nothing, or make no investment(s). This option should be included in every analysis. In other cases, it may be possible to make a number of alternative, independent investments in parallel.

Several methods exist for selecting the best alternative from a group of proposals.

- Present Worth
- Capitalized Cost
- Annual Cost
- Benefit-Cost Ratio
- Rate of Return

Each of these methods have their own merits, faults and applications.

# 9.1 Present Worth Method

The basis for Present Worth Analyses should be based on what has been covered so far. When two or more alternatives are capable of performing the same function, the superior alternative has the largest present worth. This method is suitable for ranking alternatives, and indicates a “degree” of superiority among alternatives. However, this method does have some limitations. It is restricted to mutually exclusive alternatives, and alternatives that have the same live span.

Example

Project A

- Present Cost = $10,000 today
- Future Payback = $11,500 in two years

Project B

- Present Cost = $8,000 today
- Future Payback = $4,500/yr for two years

Comparison

- $P_A(2)$ = -10,000 + 11,500(P/F,5%,2) = $431
- $P_B(2)$ = -8,000 + 4,500 (P/A,5%,2) = $361

Conclusion?

Project A is superior.

# 9.2 Capitalized Cost Method

The Capitalized Cost of an alternative is the initial cost plus the annual sustaining cost in present dollars, or

*Capitalized Cost = Initial Cost + Annual Cost/i*

This Equation used when "Annual Cost" is equal in every year. If O&M costs occur irregularly, then their values must be converted into "Equal Annual Amounts (EAA)". The EAA is calculated by moving irregular values to "present worth" then distributing them by multiplying by the A/P discount factor. Alternatively, the following equation may be used to convert the equal annual amount back to the present worth.

*Capitalized Cost = Initial Cost + EEA/i*

This method is useful when annual costs are equal every year, and provides a suitable means for ranking alternatives. This method also indicates "degree" of superiority among alternatives. This method is applied to one or more alternatives with an "infinite" life span. It is more work when O&M costs are irregular.

# 9.3 Annual Cost Method

When two or more alternatives are capable of performing the same function, the superior alternative has the "lowest annual cost". This method assumes each alternative can be replaced by an "identical twin" at life end (infinite renewal). The calculated annual cost is known as the "Equivalent Uniform Annual Cost (EUAC)". The EUAC is positive when expenses exceed income. This method is sometimes called the "Annual Return Method" or "Capital Recovery Method".

This method represents a good approach for comparing alternatives with different life spans. It is suitable for ranking alternatives and also indicates "degree" of superiority among the alternatives. Its limitations include the restriction to mutually exclusive alternatives and that it requires alternatives to have the same project duration.

Example

Option A (Conveyor Haulage)

- Cost = $180,000 today; O&M Cost = $500/yr
- Life Span = 30 years

Option B (Truck Haulage)

- Cost = $45,000 today; O&M Cost = $2,000/yr
- Life Span = 10 years

Comparison

- $EUAC_A$ = 180,000(A/P,7%,30)+500 = $15,000
- $EUAC_B$ = 45,000(A/P,7%,10)+2,000 = $8,400

Conclusion?

Option B is superior (i.e., three trucks with life of 10 yrs and $45,000 cost will be purchased).

# 9.4 Benefit Cost Method

Benefit-Cost Method analysis identify a project as acceptable if the present worth of all benefits (regardless of the beneficiary) divided by the present worth of all costs is greater than one. It can be used to rank alternatives if an "incremental analysis" is conducted. First, it must be determined that the ratio is greater than one for each alternative. Then the alternatives can be arranged in order of lowest to highest outlay. Then the differential using (B2-B1)/(C2-C1) for each pair of alternatives is calculated. If the ratio exceeds unity, alternative 2 is best; otherwise, alternative 1 is best.

Benefit-cost Analysis is especially useful in municipal projects where benefits and costs accrue to various segments of a community, however; it may be difficult to determine whether a cash flow is cost (disbursement by sponsors) or disbenefit (disbursement by users). This method does not allow a direct ranking between competing projects. It is acceptable when benefit-cost is greater than one, regardless of how cash flow is placed, yet the numerical magnitude of the ratio depends on the placement of cash flow.

Example

Project A

- Present Cost = \$10,000 today
- Future Payback = \$11,500 in two years

Project B

- Present Cost = \$8,000 today
- Future Payback = \$4,500/yr for two years

Project A: $C_A = \$10{,}000;\ B_A = \$11{,}500(P/F,5\%,2) = \$10{,}431$

$Ratio = 10{,}431/10{,}000 = 1.04 > 1$ *{Do it!}*

Project B: $C_B = \$8{,}000;\ B_B = \$4{,}500(P/A,5\%,2) = \$8{,}367$

$Ratio = 8{,}367/8{,}000 = 1.05 > 1$ *{Do it!}*

When performing an Incremental Analysis Project B should be listed as alternative 1 since it has the lowest outlay. Project A is therefore listed as alternative 2.

- $B_2$-$B_1$ = 10,431 - 8,367 = 2064
- $C_2$-$C_1$ = 10,000 - 8,000 = 2000
- $(B_2-B_1)/(C_2-C_1)$ = 2064/2000 = 1.03

Conclusion?

Since the Incremental Ratio exceeds unity, Alternative 2 (Project A) should be selected.

Note – This particular project had a poorer benefit-cost ratio (don't use this for rankings).

# 9.5 Rate of Return Method

Projects with a rate of return (ROR) exceeding the minimum attractive rate of return (MARR) are attractive as investments. ROR is defined as the interest rate that will discount all cash flows to a total present worth equal to the initial required investment. Calculating ROR by dividing annual receipts (returns) by the initial investment is common, but incorrect. This ignores time value of money and items such as salvage, depreciation, taxes, etc.

The procedure for calculating ROR is accomplished iteratively (Trail-and-Error):

- Calculate present worth using an arbitrary, but reasonable, value for i.
- Choose another value for i and again compute the present worth.
- Extrapolate to find the value of i (ROR) that gives a zero present worth.
- May need several points to improve accuracy.

Figure 9.1 shows a plot of Present Worth versus interest rates.

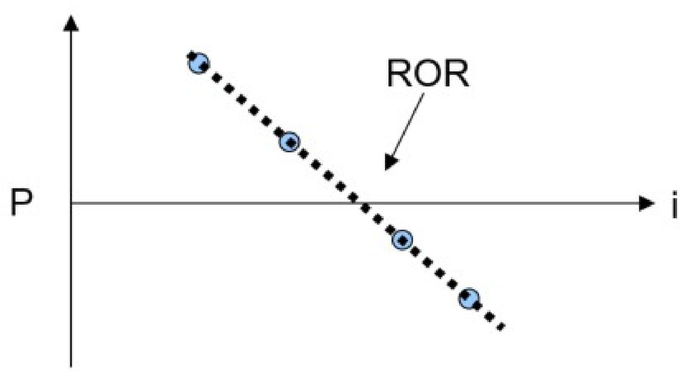



**Figure 9.1: ROR Plot.**

ROR requires an “incremental analysis” for mutually exclusive alternatives, also commonly called “rate of return on added investment study”. The procedure starts by subtracting annual cash flows for an alternative with lower initial cost from those of an alternative with a higher initial cost. This produces a third alternative representing the costs and benefits of added investment. The added expense of higher cost investment is not warranted unless ROR for this third alternative also exceeds the MARR.

The primary advantage is that no knowledge of interest rate is necessary, however; this method if often poorly understood and misapplied. It is not possible to simply select the project with the highest ROR as the best alternative (i.e., can’t use for ranking).

Example

Project A

- Present Cost = $10,000 today
- Future Payback = $11,500 in two years

Project B

- Present Cost = $8,000 today
- Future Payback = $4,500/yr for two years

Comparing the ROR values:

$P_A(2) = -10,000 + 11,500(P/F,i\%,2) = \$0$ ROR = 7.2%

$P_B(2) = -8,000 + 4,500\ (P/A,i\%,2) = \$0$ ROR = 8.2%

Both are attractive if the MARR = 5%.

$P_{B\text{-}A} = -2,000-4,500(P/F,i\%,1)+7,000(P/F,i\%,2) = \$0$

ROR = 5.8%

In this case, option A should be chosen – Spending an extra $2,000 gives an ROR greater than the MARR.

Companies identify numerous opportunities to spend money that will generate future returns, each with an expected rate of return. Companies may identify numerous sources of funds that can be used for investments, each with an associated interest rate, or if a loan, interest charged by lender. For these scenarios, the rate is known. If funds are generated by the company, the borrowed amounts may be assigned a “fictitious” rate. In this case, the rate is unknown, but can be set comparable to other opportunities that may compete for investment dollars.

Value of a project is maximized if the ROR of all investments exceed the highest interest rate charged for borrowed funds. Every opportunity is taken to invest at an ROR exceeding that for which money can be borrowed (i.e., MARR). No investments are made when ROR < MARR. No loans are taken that charge interest rates > MARR. Generally, companies will exploit all opportunities to borrow at < MARR and invest at > MARR (after factoring risk). Determining a precise MARR requires a crystal ball. If the MARR is too low there will be inadequate funds for investment. If the MARR is too high then profitable investments will be rejected.

# 10.0 Varied Life Alternatives

Another non-standard engineering economic analysis involves “varied life alternatives.” So far, we have only compared alternatives with equal life spans. What if this is not the case?

Consider

- Fan A = 3 yrs; Fan B = 5 yrs
- If you buy Fan A, but need a fan for 5 yrs, what happens at year 3?

We must distinguish between length of the need and life span of the alternatives.

- The Time of the need is called the “analysis horizon”.
- The life span(s) of the investment do not have to be the same as the horizon.

If the 5-year asset is chosen, then for 3-year analysis you need:

- Disposition (salvage) value of the alternative at t=3.
- The fact that asset is sold when it has remaining useful life does not affect analysis horizon.

If 3-year asset is chosen for the 5-year analysis you need:

- To know how the need will be satisfied for last two years.
- This may include a second purchase with salvage value, rental unit, contract unit, other…?

In either case, all costs must be established and “converted” to the same life span (then use previously discussed analysis methods).

It is also quite common to have a long-term need that is met with multiple short-lived assets. If these shorter lives represent an integer multiple, any alternative selection criteria can be used to identify the superior alternative.

- 12-yr need – four 3-yr trucks, or
- 12-yr need – three 4-yr trucks?

If present worth is used, all alternatives must be evaluated over whole horizon. If EUAC used, it is possible to calculate the annual cost on one lifespan of each alternative.

If the need horizon is infinite, it is not necessary to restrict asset lives to alternatives with integer multiples. The superior alternative will be “replaced” whenever necessary, forever. This is almost always solved with either:

- Annual cost method
- Capitalized cost method

It is common in these scenarios to assume that the cost and cash flow structure of asset replacements (renewals) are same as original asset.

“Opportunity cost” represents what is not received when an alternative is rejected. While “opportunity cost" is an imaginary cost, it is very important to include in an economic analysis. Note that not realizing salvage value can represent a lost opportunity cost. Similarly, the difference in salvage value based on a difference in salvage timeframe may also represent an lost opportunity cost.

# 10.1 Replacement Studies

A "replacement study" examines whether an existing unit should be retired. Practically, this is just an economic comparison between alternatives:

- Defender = Existing Unit
- Challenger = New/Replacement Unit

Be careful when executing this type of analysis. Although it seems logical to use the salvage value of the defender to reduce cost of challenger- by convention, the defender's salvage value is subtracted from defender's present value (keeps the costs/benefits with defender). Ultimately, the salvage value is treated as an opportunity cost incurred if the defender is not retired.

If the Defender/Challenger have the same/similar life-spans:

- Present worth analysis is used to identify superior alternative.
- The placement of salvage value has no effect on the net difference between present worth(s).
  - Although values of present worth will be different depending on placement, the difference in present worths will be the same.

If the Defender/Challenger will have different life-spans

- The annual cost comparison is used to identify the superior alternative.
- Standard conventions must be utilized since salvage value would be spread over a different number of years.
  - Must keep salvage value with defender.

In this case, it is best to calculate the cost of keeping the defender for one more year. In addition to O&M costs, this now includes:

- The opportunity interest cost incurred by not selling the defender.
- The drop in salvage value if the defender is kept for one additional year.

Thus, the defender EUAC is:

- EUAC = next year's O&M costs, plus i (current salvage value), plus the current salvage minus the next year's salvage.

It is important not to count the salvage value more than once. It is common, but incorrect, to add salvage value to the defender and also to subtract it from the challenger. The EUAC equation for the defender contains a difference in salvage value between two consecutive years. The defender/challenger decision must be made on year-by-year basis.

- One application of the equation does not mean that the defender should stay in service.
- The calculation is repeated as long as there is a drop in salvage value from year to year.

Annual O&M costs tend to increase as the unit ages. The amortized cost of a capital purchase decreases over as more years pass. Therefore, a trade-off exists between O&M costs (which increase with age) and amortized capital costs (which decrease with age). As a result, a minimum EUAC occurs that defines the “economic life” of the unit. This is referred to as the “Minimum Cost Retirement”.

# 11.0 Risk

Up to this point, our economic analyses assumed inputs are known with certainty. In reality, inputs are just "best guesses" of expected values. For example, consider two projects both with the same ROR and payback period. The first is a coal mine expansion in Kentucky, while the second is a new undersea gold mining operation off the coast of North Carolina. It is doubtful a company would view these projects equally due to significant differences in the perceived risk associated with each one.

In general, Risk can be defined as:

- Danger – The possibility of incurring loss, damage, injury or other misfortune.
- Gambling – A venture undertaken that creates exposure to possible chance of loss.

In the context of Engineering Economics, Risk can be further explained as an unforeseen deviation of individual cash flows from expected values for a capital project.

- The deviation may be positive (better) or negative (worse).
- Consider, most input values are just "best estimates" of expected values.
- These uncertainties include grade, reserves, operating costs, market prices, etc.

Wise decision-makers look at economic analyses and ask "what if" questions.

- What if production cost is higher?
- What if construction costs overrun estimates?
- What if prices fall lower than expected?
- What is the effect of a higher royalty rate?

Prior to the advent of computers, answers to such questions were difficult and required many laborious calculations and lots of time. Today, computing tools are used to easily and quickly answer such questions.

A risk analysis diagram is shown on Figure 11.1.

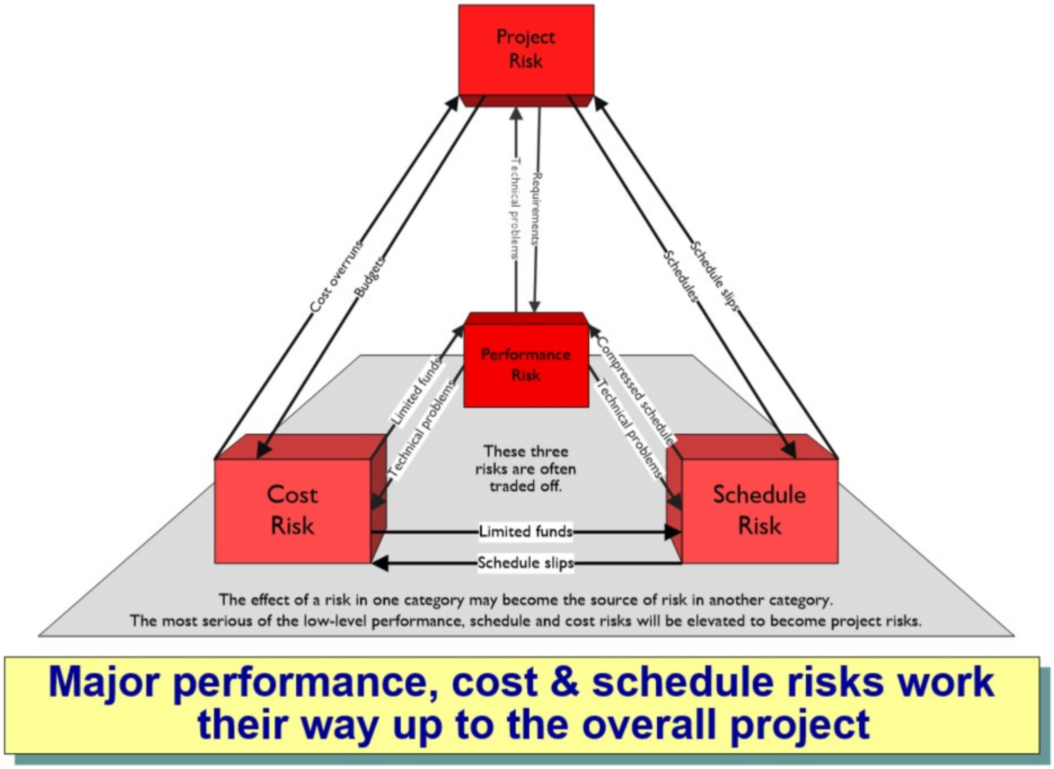



**Figure 11.1: Risk Analysis Diagram.**

Figure 11.2 shows a common risk tracking plot of the likelihood of an even occurring versus the potential severity of the consequences of that event.

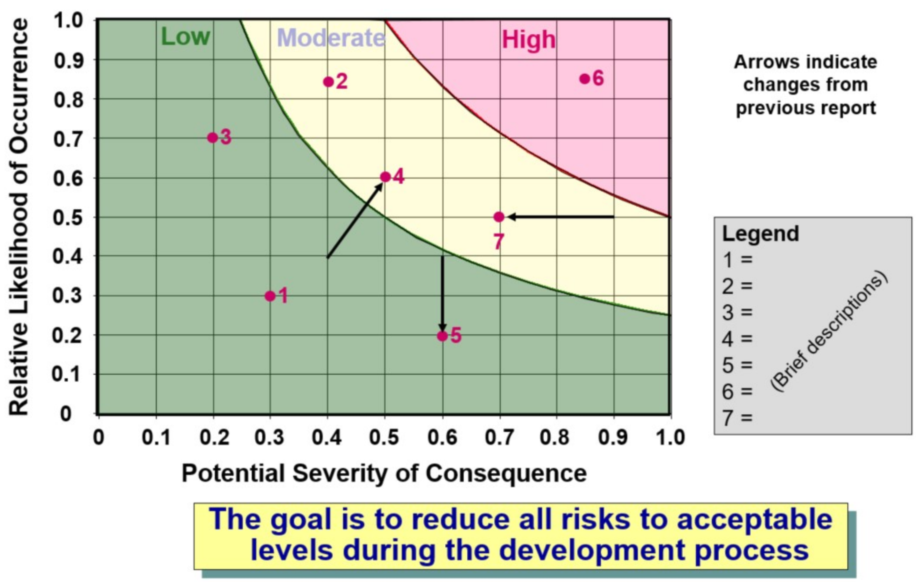



**Figure 11.2: Plot of Event Likelihood vs. Severity.**

Note that the ordinate (vertical axis) is not a “probability”. Probabilities cannot be calculated in many cases because of uncertainty and unknown unknowns. As such, the ordinate axis is labeled “relative likelihood”. Relative because it concerns relationships between risks and Likelihood not in the mathematical sense, but in the dictionary sense, indicating risks those risks that are most likely.

# 12.0 Risk Accounting

Some of the common methods used in the mining industry to “account” for risk include:

- Risk-Adjusted Payback
- Risk-Adjusted MARR
- Risk-Adjusted Inputs

Risk-Adjusted Payback is one way to compensate for risk is to demand a shorter payback period. Generally, less uncertainty is associated with near term activities. This case, cash flows are adjusted by appropriately scheduling project activities. Some potential shortcomings of this method are that it does not measure risk directly, and that it often requires subjective/arbitrary decisions. In general, a shorter payback period usually does not provide the highest overall worth, e.g., may “high-grade” a mine, thereby reducing the overall property worth.

Another way to compensate for risk is to utilize different MARR levels. Riskier projects are required to meet higher MARR values. For example, the replacement of equipment requires a much lower MARR than opening a new mine. Some potential shortcomings of this method are that it requires subjective hurdle values, and can lead to inconsistency in results since MARR assignments are “fuzzy”.

One additional way to compensate for risk is to adjust the input values. This may reduce subjectivity by quantifying the “range” over which input values may vary. This generally requires the effective utilization of statistical tools (i.e., “probability analyses”); however, assigning all “worst case” scenarios may result in rejecting a profitable project. Mine operators are often conservative with their inputs to ensure a “safe” investment, and low market prices often used in mining, result in many rejected projects.

Methods for mathematically “quantifying” risk include:

- Sensitivity Analysis
- Probability Analysis
- Statistical Analysis

Both methods seek to predict the outcome of a decision if a situation turns out to be different compared to the initial forecasts and study how variation in inputs can be apportioned, qualitatively or quantitatively, to expected output.

# 12.1 Sensitivity Analysis

Sensitivity Analysis is the determination of the response of profitability indicators to changes in estimated factors. Sensitivity analysis helps to show input factors that are major drivers to unfavorable economics. The approach does not provide a measure of risk – other tools are still needed to "quantify" financial risk.

First, an economic model is set up to describe the investment under consideration. One factor at a time is then changed (often by a fixed amount such as 10%). Finally, how the profitability indicator responds to the change is examined.

Sensitivity analysis provides a qualitative measure of "risk", but:

- When and where does the return go below an acceptable level?
- If the profit indicator does not react to a factor change, the uncertainty related to the estimation of that particular factor is reduced (or vice-versa).

The method does not, however, quantify the "likelihood" that a poor return will occur. This can be handled by probability analysis, which will be further discussed in another section.

For tabular presentation of results, a table (or matrix) showing the factors of interest as a function of one or more profitability indicators is constructed. The table can have multiple input factors and profitability indicators. Typically, this is reported as a percentage change in response for a fixed percentage change for all inputs. For example, a 10% increase in opex produced a 26% decrease in ROR, or a 10% increase in capex produced a 5.2% decrease in ROR, etc.

An example of a tabular Sensitivity Analysis is shown on Figure 12.1.

Effect of increasing input factor by 10%.

| Input Factor | DCF-ROR (%) | Payback Period (%) |
|---|---|---|
| Capex | -5.2 | -4.2 |
| Opex | -25.6 | -19.6 |
| Tax | -17.2 | -12.3 |
| Salvage | +7.6 | +3.2 |

**Figure 12.1: Tabular Sensitivity Analysis Example.**

In order to represent the results of a Sensitivity Analysis graphically, the chosen profitability indicator as a function of factor of interest is plotted, e.g., a plot of ROR vs. market price. This may include other factors in the plot, so long as the relevant information is clear. A steep curve indicates a sensitive parameter. This method may also include a variation in which “percentage unfavorable change” for each factor is plotted as a function of the profitability indicator. Graphical analysis allows many different factors to be plotted on a single x-axis.

An example of graphical Sensitivity Analysis is shown on Figure 12.2:

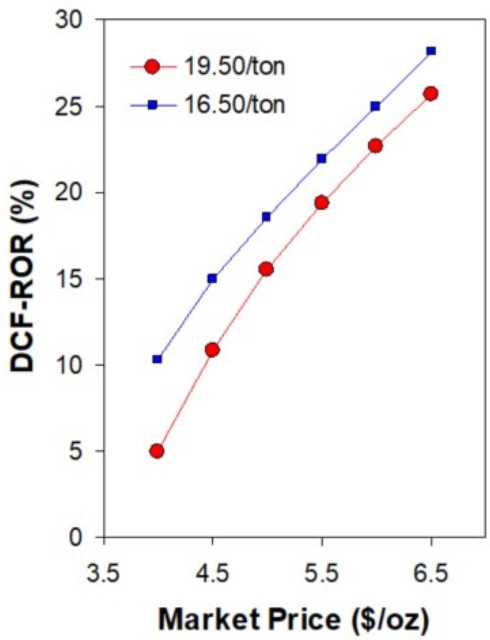
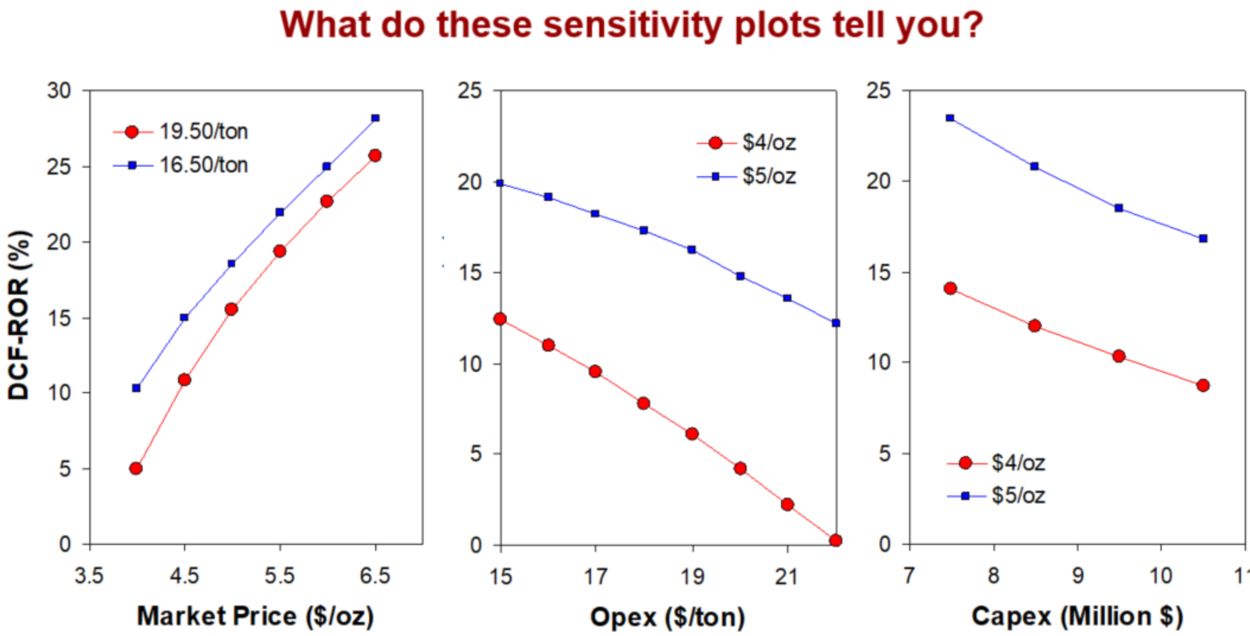



**Figure 12.2: Example of Graphical Sensitivity Analysis.**




In mining ventures, “sensitive” parameters typically include:

- metal or mineral price
- operating cost
- annual production grade
- initial capital investment

Sensitivity analysis becomes “probability analysis” when occurrence probability can be associated with parameter levels.  This allows the risk factor to be quantified in terms of total probability (i.e., quantifying risk).  In some ways, this is similar to predicting the likelihood of a given hand in poker.

# 13.0 Probability Analysis

Probability Analysis establishes risk level and magnitude by defining probability distributions for input values rather than treating each as fixed, e.g., market price may be assigned a range of values with specific occurrence probabilities. This is a subset of the risk-adjusted input method.

Probability distributions can be “discrete” or “continuous”. Discrete distributions factors are assigned a set of finite values and probabilities. Continuous distributions factors are assigned a continuous range of values and probabilities.

Consider a coin flip that can give either “heads” or “tails” as possible outcomes. Then consider, “what is the probability that the flipped coin will come up as heads?” How many iterations will be required to obtain a “good” value?

What is the probability that the flipped coin will:

- come up as “heads” on seven out of ten attempts?
- Come up as “heads” on at least seven out of ten attempts?

**Mathematical Solution:**

10 tosses = 1024 possible outcomes (i.e., 210)

In 10 tosses, 120 have exactly 7 heads (p=120/1024=11.7%)

In 10 tosses, 176 have 7, 8, 9 or 10 heads (p=176/1024=17.2%)

Now consider the case of evaluating the worth of an investment in a small copper mine, with the following known values:

- Ore Grade = 0.8%
- Mill Recovery = 90%
- Mine Life = 10 years
- Initial Capital Investment = $67.5 million
- Operating Period = 350 days/year
- Copper Price = $2.50 / lb
- Annual Operating Cost = $17.5 million (including taxes)
- To simplify, please ignore royalties, book deductions, salvage values, etc.

The “baseline” for this evaluation is:

- Production = 0.8% x 3000 ton/day x 350 days/yr x 2000 lb/ton x 0.9 = $15.12 \times 10^6$ lb
- Gross Income = $15.12 \times 10^6$ lb x $2.50/lb = $37.8 million
- Annual Net Cash Inflow = $37.8 - $17.5 = $20.3 million
- P = A [P/A, i%, n] $67.5 = $20.3 [P/A, i%, 10]

- i = 27% ROR {Baseline Scenario}

But does this represent a good investment…?

Since estimates of inputs may be imperfect, probabilities can be assigned to each.

Mill Recovery

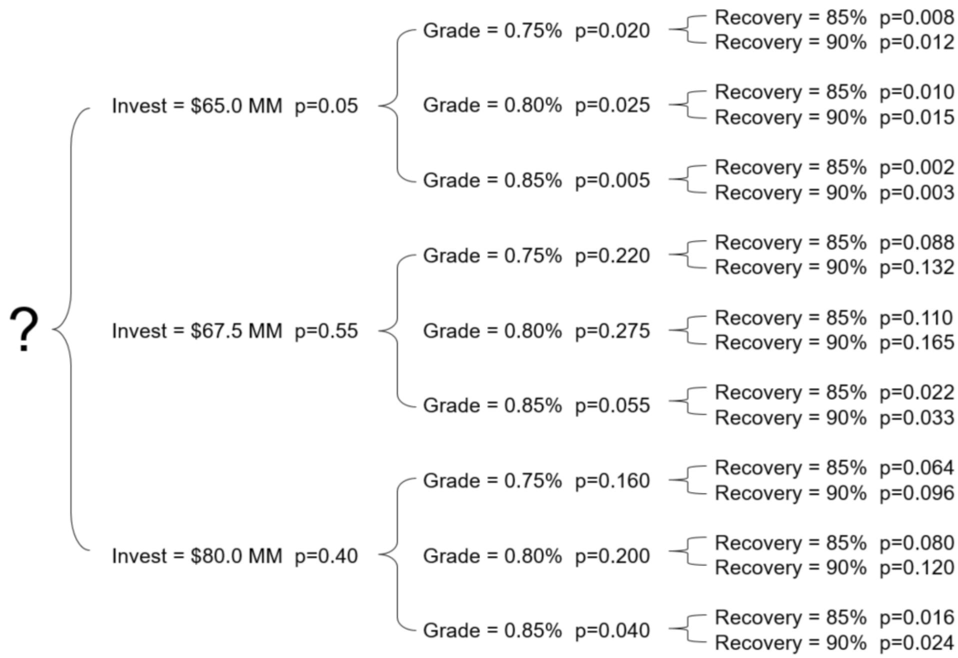
- 60% probability = 90%
- 40% probability = 85%

Ore Grade

- 40% probability = 0.75%
- 50% probability = 0.80%
- 10% probability = 0.85%

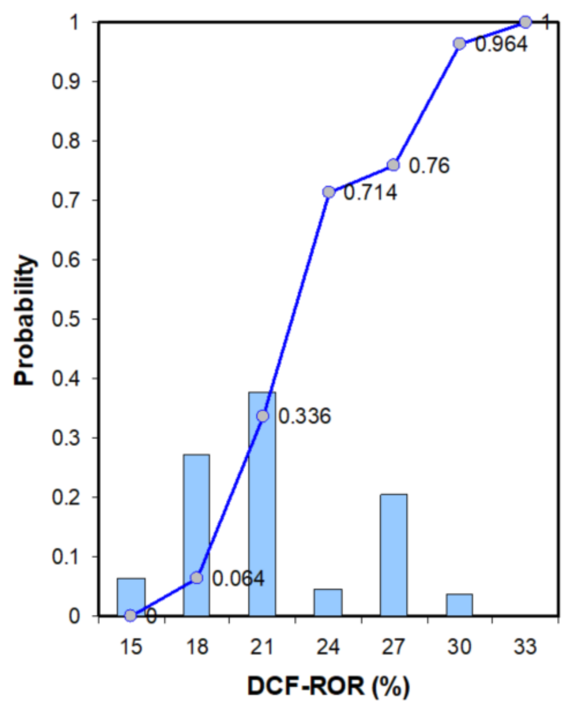
Capital Investment

- 5% probability = $65 million
- 55% probability = $67.5 million
- 40% probability = $80 million




| ROR | | Class | Cum. Probability | |
|---|---|---|---|---|
| Lower (%) | Upper (%) | Prob. -- | Exceed Lower | Below Upper |
| -- | 15 | 0.000 | -- | 0.000 |
| 15 | 18 | 0.064 | 1.000 | 0.064 |
| 18 | 21 | 0.272 | 0.936 | 0.336 |
| 21 | 24 | 0.378 | 0.664 | 0.714 |
| 24 | 27 | 0.046 | 0.286 | 0.760 |
| 27 | 30 | 0.204 | 0.240 | 0.964 |
| 30 | 33 | 0.036 | 0.036 | 1.000 |
| 33 | -- | 0.000 | 0.000 | -- |

*Represents likelihood ROR will exceed lower or upper limits (i.e., 93.6% chance ROR>18% and 33.6% ROR<21%).



| | | Investment $65.0 MM | | Investment $67.5 MM | | Investment $80.0 MM | |
|---|---|---|---|---|---|---|---|
| Mill Rec. (%) | Ore Grade (%) | ROR (%) | Prob. Value -- | ROR (%) | Prob. Value -- | ROR (%) | Prob. Value -- |
| 85 | 0.75 | 20.9 | 0.008 | 19.8 | 0.088 | 15.0 | 0.064 |
| 85 | 0.80 | 25.0 | 0.010 | 23.0 | 0.110 | 18.5 | 0.080 |
| 85 | 0.85 | 29.0 | 0.002 | 27.6 | 0.022 | 22.1 | 0.016 |
| 90 | 0.75 | 24.5 | 0.012 | 23.3 | 0.132 | 18.2 | 0.096 |
| 90 | 0.80 | 28.7 | 0.015 | 27.4 | 0.165 | 21.9 | 0.120 |
| 90 | 0.85 | 32.8 | 0.003 | 31.4 | 0.033 | 25.4 | 0.024 |

Note that:

- 3% fall within m ± s
- 5% fall within m ± 2s
- 7% fall within m ± 3s

A Normal Distribution plot is shown on Figure 13.1.

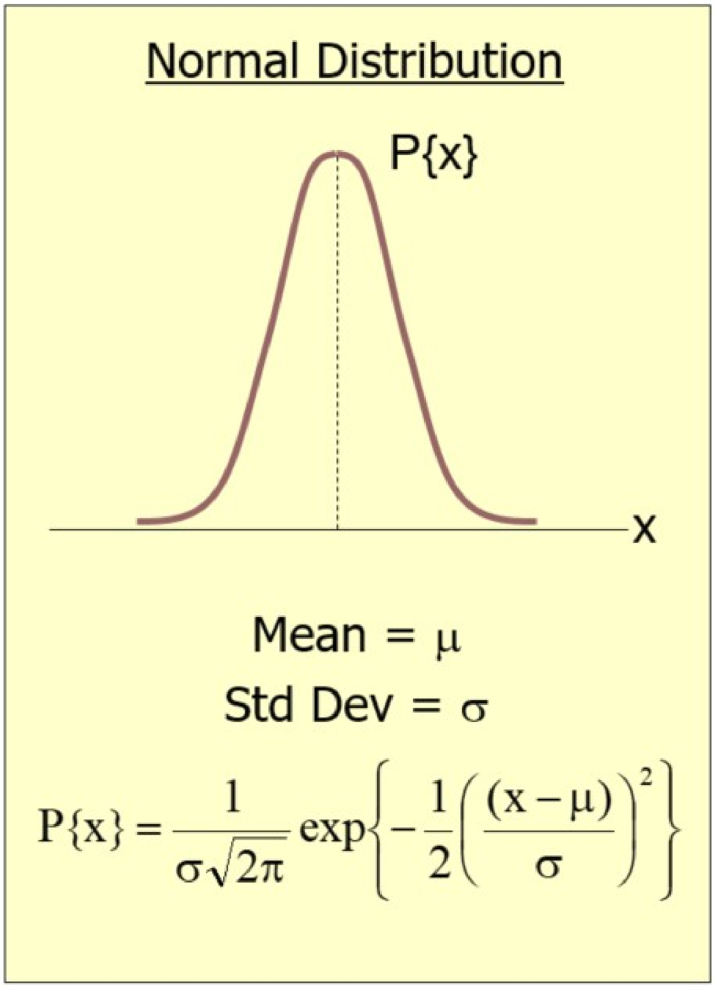



**Figure 13.1: Normal Distribution Plot.**

Figure 13.2 shows common variations in normal distribution plots.

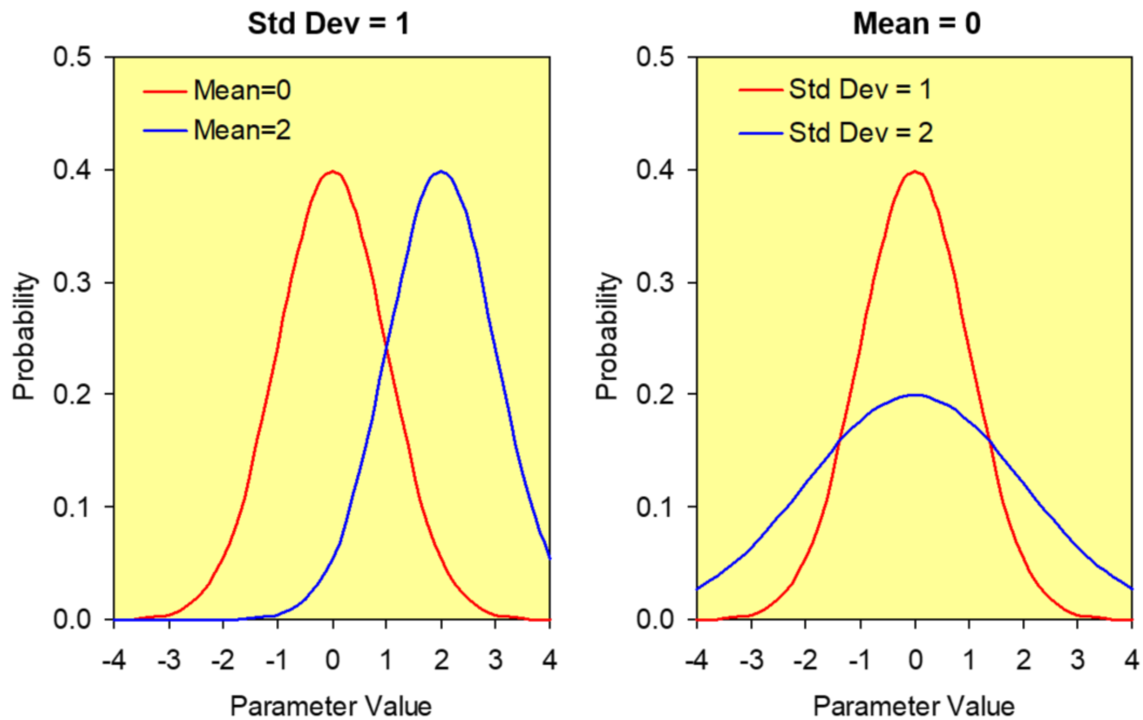



**Figure 13.2: Common Variations in normal distribution plots.**

A normal distribution with m=0 and s=1 is called the "standard normal" distribution. Z-statistic converts "normal" to "standard normal" distribution.

$$Z = \frac{x - \mu}{\sigma}$$

Values are often listed in standard tables.

Figure 13.3 shows a standard normal distribution.

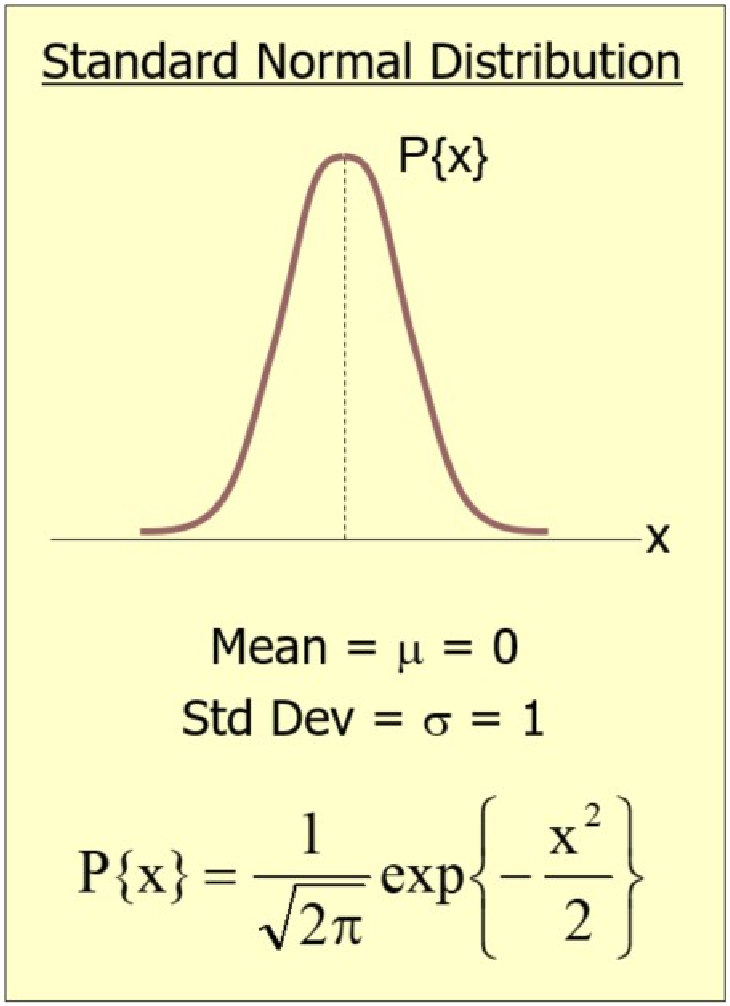
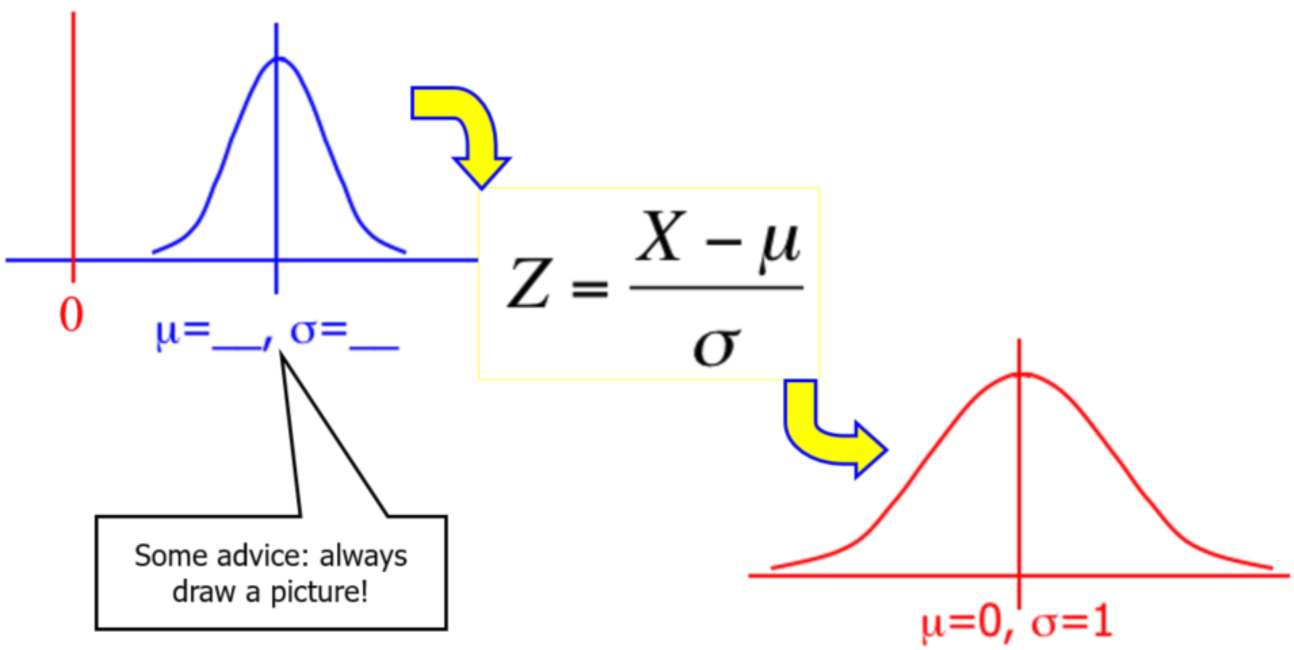
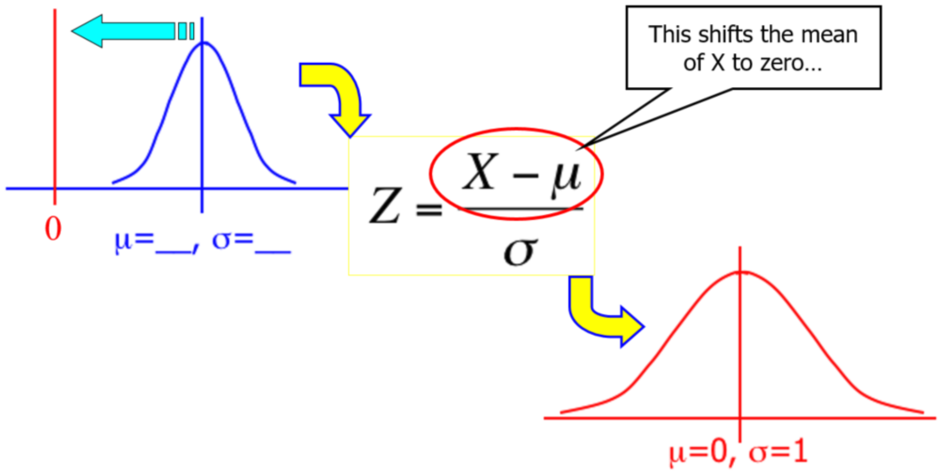



**Figure 13.3: Standard Normal Distribution.**

The Z-statistic converts any normal random variable to a standard normal random variable.


0
μ=__, σ=__
$Z = \frac{X - \mu}{\sigma}$
Some advice: always draw a picture!
μ=0, σ=1



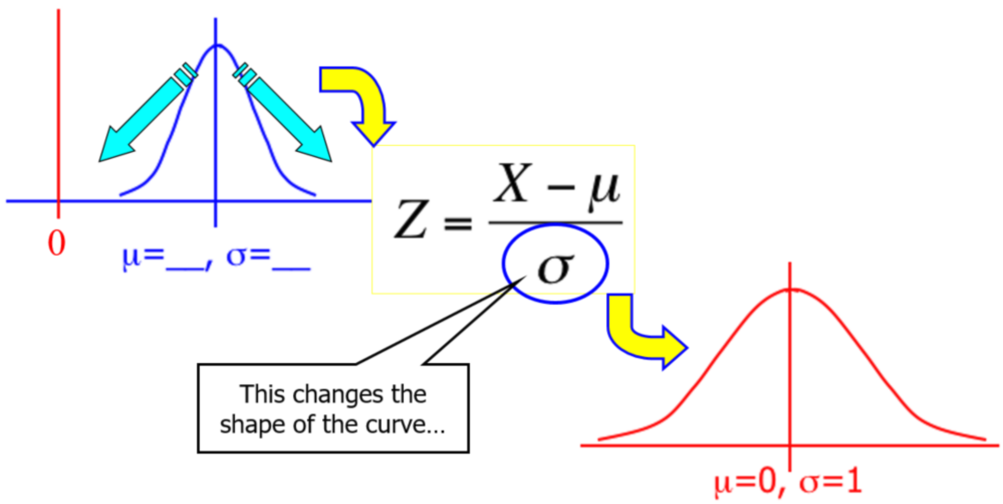




Table 13.1 gives the Z-Statistic Probability for Z-values between 0.0 and 3.5 (Points on Curve).

**Table 13.1: Z-Statistic Probability.**

| Z-STATISTIC PROBABILITIES (AREA UNDER CURVE) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| Z | 0.00 | 0.01 | 0.02 | 0.03 | 0.04 | 0.05 | 0.06 | 0.07 | 0.08 | 0.09 |
| 0.0 | 0.5000 | 0.5040 | 0.5080 | 0.5120 | 0.5160 | 0.5199 | 0.5239 | 0.5279 | 0.5319 | 0.5359 |
| 0.1 | 0.5398 | 0.5438 | 0.5478 | 0.5517 | 0.5557 | 0.5596 | 0.5636 | 0.5675 | 0.5714 | 0.5753 |
| 0.2 | 0.5793 | 0.5832 | 0.5871 | 0.5910 | 0.5948 | 0.5987 | 0.6026 | 0.6064 | 0.6103 | 0.6141 |
| 0.3 | 0.6179 | 0.6217 | 0.6255 | 0.6293 | 0.6331 | 0.6368 | 0.6406 | 0.6443 | 0.6480 | 0.6517 |
| 0.4 | 0.6554 | 0.6591 | 0.6628 | 0.6664 | 0.6700 | 0.6736 | 0.6772 | 0.6808 | 0.6844 | 0.6879 |
| 0.5 | 0.6915 | 0.6950 | 0.6985 | 0.7019 | 0.7054 | 0.7088 | 0.7123 | 0.7157 | 0.7190 | 0.7224 |
| 0.6 | 0.7257 | 0.7291 | 0.7324 | 0.7357 | 0.7389 | 0.7422 | 0.7454 | 0.7486 | 0.7517 | 0.7549 |
| 0.7 | 0.7580 | 0.7611 | 0.7642 | 0.7673 | 0.7704 | 0.7734 | 0.7764 | 0.7794 | 0.7823 | 0.7852 |
| 0.8 | 0.7881 | 0.7910 | 0.7939 | 0.7967 | 0.7995 | 0.8023 | 0.8051 | 0.8078 | 0.8106 | 0.8133 |
| 0.9 | 0.8159 | 0.8186 | 0.8212 | 0.8238 | 0.8264 | 0.8289 | 0.8315 | 0.8340 | 0.8365 | 0.8389 |
| 1.0 | 0.8413 | 0.8438 | 0.8461 | 0.8485 | 0.8508 | 0.8531 | 0.8554 | 0.8577 | 0.8599 | 0.8621 |
| 1.1 | 0.8643 | 0.8665 | 0.8686 | 0.8708 | 0.8729 | 0.8749 | 0.8770 | 0.8790 | 0.8810 | 0.8830 |
| 1.2 | 0.8849 | 0.8869 | 0.8888 | 0.8907 | 0.8925 | 0.8944 | 0.8962 | 0.8980 | 0.8997 | 0.9015 |
| 1.3 | 0.9032 | 0.9049 | 0.9066 | 0.9082 | 0.9099 | 0.9115 | 0.9131 | 0.9147 | 0.9162 | 0.9177 |
| 1.4 | 0.9192 | 0.9207 | 0.9222 | 0.9236 | 0.9251 | 0.9265 | 0.9279 | 0.9292 | 0.9306 | 0.9319 |
| 1.5 | 0.9332 | 0.9345 | 0.9357 | 0.9370 | 0.9382 | 0.9394 | 0.9406 | 0.9418 | 0.9429 | 0.9441 |
| 1.6 | 0.9452 | 0.9463 | 0.9474 | 0.9484 | 0.9495 | 0.9505 | 0.9515 | 0.9525 | 0.9535 | 0.9545 |
| 1.7 | 0.9554 | 0.9564 | 0.9573 | 0.9582 | 0.9591 | 0.9599 | 0.9608 | 0.9616 | 0.9625 | 0.9633 |
| 1.8 | 0.9641 | 0.9649 | 0.9656 | 0.9664 | 0.9671 | 0.9678 | 0.9686 | 0.9693 | 0.9699 | 0.9706 |
| 1.9 | 0.9713 | 0.9719 | 0.9726 | 0.9732 | 0.9738 | 0.9744 | 0.9750 | 0.9756 | 0.9761 | 0.9767 |
| 2.0 | 0.9772 | 0.9778 | 0.9783 | 0.9788 | 0.9793 | 0.9798 | 0.9803 | 0.9808 | 0.9812 | 0.9817 |
| 2.1 | 0.9821 | 0.9826 | 0.9830 | 0.9834 | 0.9838 | 0.9842 | 0.9846 | 0.9850 | 0.9854 | 0.9857 |
| 2.2 | 0.9861 | 0.9864 | 0.9868 | 0.9871 | 0.9875 | 0.9878 | 0.9881 | 0.9884 | 0.9887 | 0.9890 |
| 2.3 | 0.9893 | 0.9896 | 0.9898 | 0.9901 | 0.9904 | 0.9906 | 0.9909 | 0.9911 | 0.9913 | 0.9916 |
| 2.4 | 0.9918 | 0.9920 | 0.9922 | 0.9925 | 0.9927 | 0.9929 | 0.9931 | 0.9932 | 0.9934 | 0.9936 |
| 2.5 | 0.9938 | 0.9940 | 0.9941 | 0.9943 | 0.9945 | 0.9946 | 0.9948 | 0.9949 | 0.9951 | 0.9952 |
| 2.6 | 0.9953 | 0.9955 | 0.9956 | 0.9957 | 0.9959 | 0.9960 | 0.9961 | 0.9962 | 0.9963 | 0.9964 |
| 2.7 | 0.9965 | 0.9966 | 0.9967 | 0.9968 | 0.9969 | 0.9970 | 0.9971 | 0.9972 | 0.9973 | 0.9974 |
| 2.8 | 0.9974 | 0.9975 | 0.9976 | 0.9977 | 0.9977 | 0.9978 | 0.9979 | 0.9979 | 0.9980 | 0.9981 |
| 2.9 | 0.9981 | 0.9982 | 0.9982 | 0.9983 | 0.9984 | 0.9984 | 0.9985 | 0.9985 | 0.9986 | 0.9986 |
| 3.0 | 0.9987 | 0.9987 | 0.9987 | 0.9988 | 0.9988 | 0.9989 | 0.9989 | 0.9989 | 0.9990 | 0.9990 |
| 3.1 | 0.9990 | 0.9991 | 0.9991 | 0.9991 | 0.9992 | 0.9992 | 0.9992 | 0.9992 | 0.9993 | 0.9993 |
| 3.2 | 0.9993 | 0.9993 | 0.9994 | 0.9994 | 0.9994 | 0.9994 | 0.9994 | 0.9995 | 0.9995 | 0.9995 |
| 3.3 | 0.9995 | 0.9995 | 0.9995 | 0.9996 | 0.9996 | 0.9996 | 0.9996 | 0.9996 | 0.9996 | 0.9997 |
| 3.4 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9998 |
| 3.5 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 |

Table 13.2 gives the Z-Statistic Probability for Z-values between 0.0 and 3.5 (Points on Curve).

**Table 13.2: Z-Statistic Probability.**

| Z-STATISTIC PROBABILITIES (AREA UNDER CURVE) | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| Z | 0.00 | 0.01 | 0.02 | 0.03 | 0.04 | 0.05 | 0.06 | 0.07 | 0.08 | 0.09 |
| 0.0 | 0.5000 | 0.5040 | 0.5080 | 0.5120 | 0.5160 | 0.5199 | 0.5239 | 0.5279 | 0.5319 | 0.5359 |
| 0.1 | 0.5398 | 0.5438 | 0.5478 | 0.5517 | 0.5557 | 0.5596 | 0.5636 | 0.5675 | 0.5714 | 0.5753 |
| 0.2 | 0.5793 | 0.5832 | 0.5871 | 0.5910 | 0.5948 | 0.5987 | 0.6026 | 0.6064 | 0.6103 | 0.6141 |
| 0.3 | 0.6179 | 0.6217 | 0.6255 | 0.6293 | 0.6331 | 0.6368 | 0.6406 | 0.6443 | 0.6480 | 0.6517 |
| 0.4 | 0.6554 | 0.6591 | 0.6628 | 0.6664 | 0.6700 | 0.6736 | 0.6772 | 0.6808 | 0.6844 | 0.6879 |
| 0.5 | 0.6915 | 0.6950 | 0.6985 | 0.7019 | 0.7054 | 0.7088 | 0.7123 | 0.7157 | 0.7190 | 0.7224 |
| 0.6 | 0.7257 | 0.7291 | 0.7324 | 0.7357 | 0.7389 | 0.7422 | 0.7454 | 0.7486 | 0.7517 | 0.7549 |
| 0.7 | 0.7580 | 0.7611 | 0.7642 | 0.7673 | 0.7704 | 0.7734 | 0.7764 | 0.7794 | 0.7823 | 0.7852 |
| 0.8 | 0.7881 | 0.7910 | 0.7939 | 0.7967 | 0.7995 | 0.8023 | 0.8051 | 0.8078 | 0.8106 | 0.8133 |
| 0.9 | 0.8159 | 0.8186 | 0.8212 | 0.8238 | 0.8264 | 0.8289 | 0.8315 | 0.8340 | 0.8365 | 0.8389 |
| 1.0 | 0.8413 | 0.8438 | 0.8461 | 0.8485 | 0.8508 | 0.8531 | 0.8554 | 0.8577 | 0.8599 | 0.8621 |
| 1.1 | 0.8643 | 0.8665 | 0.8686 | 0.8708 | 0.8729 | 0.8749 | 0.8770 | 0.8790 | 0.8810 | 0.8830 |
| 1.2 | 0.8849 | 0.8869 | 0.8888 | 0.8907 | 0.8925 | 0.8944 | 0.8962 | 0.8980 | 0.8997 | 0.9015 |
| 1.3 | 0.9032 | 0.9049 | 0.9066 | 0.9082 | 0.9099 | 0.9115 | 0.9131 | 0.9147 | 0.9162 | 0.9177 |
| 1.4 | 0.9192 | 0.9207 | 0.9222 | 0.9236 | 0.9251 | 0.9265 | 0.9279 | 0.9292 | 0.9306 | 0.9319 |
| 1.5 | 0.9332 | 0.9345 | 0.9357 | 0.9370 | 0.9382 | 0.9394 | 0.9406 | 0.9418 | 0.9429 | 0.9441 |
| 1.6 | 0.9452 | 0.9463 | 0.9474 | 0.9484 | 0.9495 | 0.9505 | 0.9515 | 0.9525 | 0.9535 | 0.9545 |
| 1.7 | 0.9554 | 0.9564 | 0.9573 | 0.9582 | 0.9591 | 0.9599 | 0.9608 | 0.9616 | 0.9625 | 0.9633 |
| 1.8 | 0.9641 | 0.9649 | 0.9656 | 0.9664 | 0.9671 | 0.9678 | 0.9686 | 0.9693 | 0.9699 | 0.9706 |
| 1.9 | 0.9713 | 0.9719 | 0.9726 | 0.9732 | 0.9738 | 0.9744 | 0.9750 | 0.9756 | 0.9761 | 0.9767 |
| 2.0 | 0.9772 | 0.9778 | 0.9783 | 0.9788 | 0.9793 | 0.9798 | 0.9803 | 0.9808 | 0.9812 | 0.9817 |
| 2.1 | 0.9821 | 0.9826 | 0.9830 | 0.9834 | 0.9838 | 0.9842 | 0.9846 | 0.9850 | 0.9854 | 0.9857 |
| 2.2 | 0.9861 | 0.9864 | 0.9868 | 0.9871 | 0.9875 | 0.9878 | 0.9881 | 0.9884 | 0.9887 | 0.9890 |
| 2.3 | 0.9893 | 0.9896 | 0.9898 | 0.9901 | 0.9904 | 0.9906 | 0.9909 | 0.9911 | 0.9913 | 0.9916 |
| 2.4 | 0.9918 | 0.9920 | 0.9922 | 0.9925 | 0.9927 | 0.9929 | 0.9931 | 0.9932 | 0.9934 | 0.9936 |
| 2.5 | 0.9938 | 0.9940 | 0.9941 | 0.9943 | 0.9945 | 0.9946 | 0.9948 | 0.9949 | 0.9951 | 0.9952 |
| 2.6 | 0.9953 | 0.9955 | 0.9956 | 0.9957 | 0.9959 | 0.9960 | 0.9961 | 0.9962 | 0.9963 | 0.9964 |
| 2.7 | 0.9965 | 0.9966 | 0.9967 | 0.9968 | 0.9969 | 0.9970 | 0.9971 | 0.9972 | 0.9973 | 0.9974 |
| 2.8 | 0.9974 | 0.9975 | 0.9976 | 0.9977 | 0.9977 | 0.9978 | 0.9979 | 0.9979 | 0.9980 | 0.9981 |
| 2.9 | 0.9981 | 0.9982 | 0.9982 | 0.9983 | 0.9984 | 0.9984 | 0.9985 | 0.9985 | 0.9986 | 0.9986 |
| 3.0 | 0.9987 | 0.9987 | 0.9987 | 0.9988 | 0.9988 | 0.9989 | 0.9989 | 0.9989 | 0.9990 | 0.9990 |
| 3.1 | 0.9990 | 0.9991 | 0.9991 | 0.9991 | 0.9992 | 0.9992 | 0.9992 | 0.9992 | 0.9993 | 0.9993 |
| 3.2 | 0.9993 | 0.9993 | 0.9994 | 0.9994 | 0.9994 | 0.9994 | 0.9994 | 0.9995 | 0.9995 | 0.9995 |
| 3.3 | 0.9995 | 0.9995 | 0.9995 | 0.9996 | 0.9996 | 0.9996 | 0.9996 | 0.9996 | 0.9996 | 0.9997 |
| 3.4 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9998 |
| 3.5 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 |

# 14.0 Statistical Analysis of Risk

The “expected profit” from a project can be calculated using:

$$E(\pi) = \bar{\pi} = \sum_{i=1}^{n} \pi_i (P_i)$$

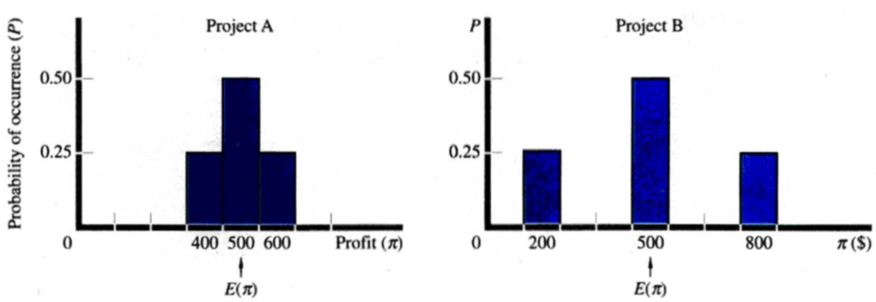
This requires the knowledge of the relative probabilities that various events will occur. Consider two projects with expected profit of $500…

| Project | State of Economy | Probability (P) | Outcome ($\pi$) | Expected Value |
|---|---|---|---|---|
| A | Boom | 0.25 | $600 | $150 |
| | Normal | 0.50 | 500 | 250 |
| | Recession | 0.25 | 400 | 100 |
| | Expected profit from Project A | | | $500 |
| B | Boom | 0.25 | $800 | $200 |
| | Normal | 0.50 | 500 | 250 |
| | Recession | 0.25 | 200 | 50 |
| | Expected profit from Project B | | | $500 |

## Discrete Probability Distributions

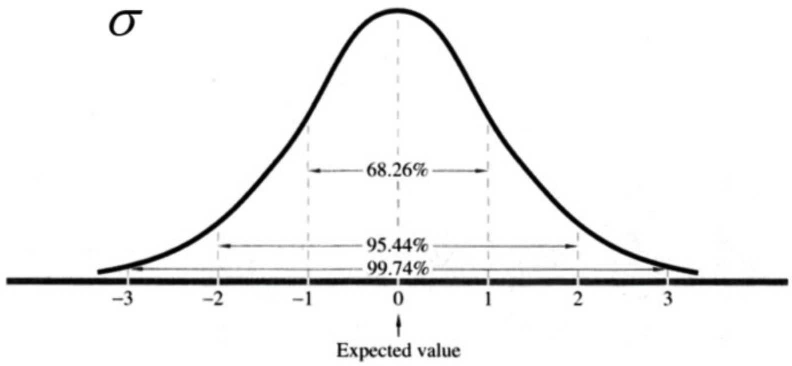
Project A; $E(\pi)$ = 500, Low Risk

Project B: $E(\pi)$ = 500, High Risk



# The Normal Distribution

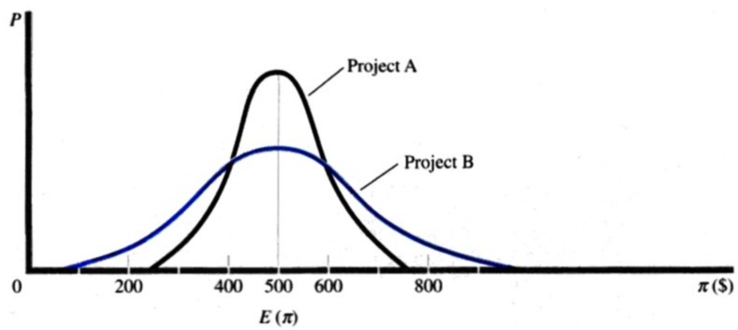
$$Z = \frac{\pi_i - \overline{\pi}}{\sigma}$$




# Continuous Probability Distributions

Project A: E($\pi$) = 500, Low Risk

Project B: E($\pi$) = 500, High Risk




The "standard deviation" provides an absolute measure of risk.

$$\sigma = \sqrt{\sum_{i=1}^{n} (X_i - \overline{X})^2 (P_i)}$$

For Project A

$$\sigma = \sqrt{0.25(600-500)^2 + 0.5(500-500)^2 + 0.25(400-500)^2}$$
$$= \$70.71$$

For Project B

$$\sigma = \sqrt{0.25(800-500)^2 + 0.5(500-500)^2 + 0.25(200-500)^2}$$
$$= \$212.13$$

The “coefficient of variation” provides a relative measure of risk.

$$\upsilon = \sigma / \bar{\pi}$$

For Project A

$$\upsilon = \sigma / \bar{\pi} = \$70.17 / 500 = 0.14$$

For Project B

$$\upsilon = \sigma / \bar{\pi} = \$212.13 / 500 = 0.42$$

This analysis suggests that Project B is about three times more risky than Project A.

# 15.0 Probabilistic Simulation

If probability distributions for input variables are known (or estimated), probabilistic simulation can be used to assess risk in economic analyses. A convenient platform for conducting such analyses is the "Monte Carlo" simulation. This method has applications in the fields of Physical Sciences and Engineering, Design and Visuals, Finance and Business, Telecommunications, Mathematics, Gaming, etc.

Monte Carlo methods are successful in risk analysis when compared with alternative methods or human intuition. For cases such as oil exploration, actual observations of failures, cost overruns and schedule overruns, results are routinely better predicted by the Monte Carlo simulations than by human intuition or alternative "soft" methods. Unfortunately, this method is widely under-utilized in the mining industry.

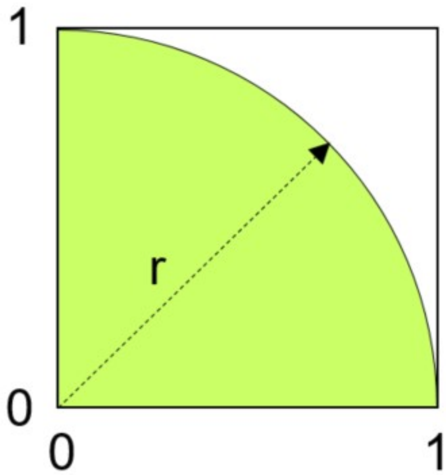
Monte Carlo algorithms rely on repeated random sampling to compute results. They are most suited to computer calculations. They are also particularly useful if it is otherwise unfeasible to find a deterministic result (i.e., not like a simple coin toss). Monte Carlo methods are particularly useful for systems with significant uncertainty in inputs (e.g., business risk, rainfall amounts, production output, etc.). The term "Monte Carlo Method" was coined in the 1940s by physicists working on nuclear weapon projects.

There is no single Monte Carlo method; instead, the term describes a large and widely-used class of approaches. However, these approaches tend to follow a particular pattern:

- Define a domain of possible inputs.
- Generate inputs randomly from the domain using a certain specified probability distribution.
- Perform a deterministic computation using the inputs.
- Aggregate the results of the individual computations into the final result.

Example

- Consider the shaded circular arc region shown for a dart board.
- What is the probability that a "dart" hits the arc region inside the square?
- What do you get if you multiply this value by 4?
- How many iterations are required to obtain a "good" value?

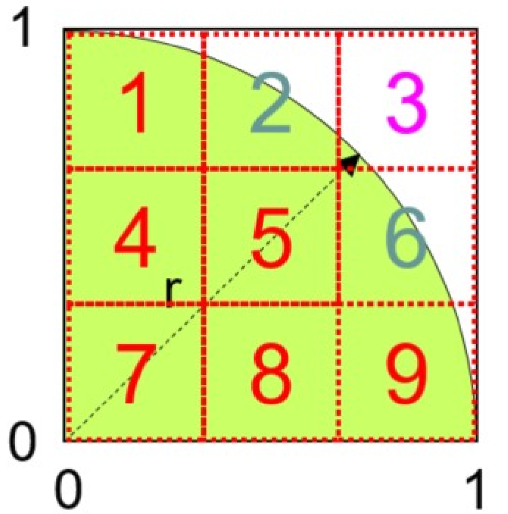



| Number Inside | Number Outside |
| --- | --- |
| | |

Inside/Total x 4 = ______

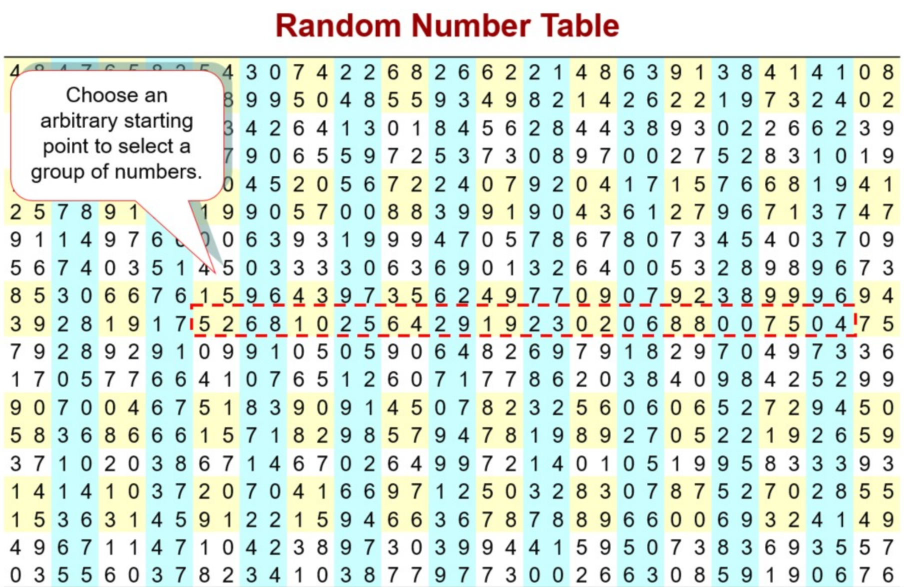
If the previous exercise is repeated, but this time a random number chart is used to act as “arrows”, what is the probability that a “number” hits the arc region inside the square?

- What do you get if you multiply this value by 4?
- How many iterations are required to obtain a “good” value?




| Number Inside | Number Outside |
|---|---|
| | |

Inside/Total x 4 = ______




4 [illegible] 4 3 0 7 4 2 2 6 8 2 6 6 2 2 1 4 8 6 3 9 1 3 8 4 1 4 1 0 8
[illegible] 8 9 9 5 0 4 8 5 5 9 3 4 9 8 2 1 4 2 6 2 2 1 9 7 3 2 4 0 2
[illegible] 3 4 2 6 4 1 3 0 1 8 4 5 6 2 8 4 4 3 8 9 3 0 2 2 6 6 2 3 9
[illegible] 7 9 0 6 5 5 9 7 2 5 3 7 3 0 8 9 7 0 0 2 7 5 2 8 3 1 0 1 9
[illegible] 0 4 5 2 0 5 6 7 2 2 4 0 7 9 2 0 4 1 7 1 5 7 6 6 8 1 9 4 1
2 5 7 8 9 1 [illegible] 1 9 9 0 5 7 0 0 8 8 3 9 9 1 9 0 4 3 6 1 2 7 9 6 7 1 3 7 4 7
9 1 1 4 9 7 6 [illegible] 0 6 3 9 3 1 9 9 9 4 7 0 5 7 8 6 7 8 0 7 3 4 5 4 0 3 7 0 9
5 6 7 4 0 3 5 1 4 5 0 3 3 3 3 0 6 3 6 9 0 1 3 2 6 4 0 0 5 3 2 8 9 8 9 6 7 3
8 5 3 0 6 6 7 6 1 5 9 6 4 3 9 7 3 5 6 2 4 9 7 7 0 9 0 7 9 2 3 8 9 9 9 6 9 4
3 9 2 8 1 9 1 7 5 2 6 8 1 0 2 5 6 4 2 9 1 9 2 3 0 2 0 6 8 8 0 0 7 5 0 4 7 5
7 9 2 8 9 2 9 1 0 9 9 1 0 5 0 5 9 0 6 4 8 2 6 9 7 9 1 8 2 9 7 0 4 9 7 3 3 6
1 7 0 5 7 7 6 6 4 1 0 7 6 5 1 2 6 0 7 1 7 7 8 6 2 0 3 8 4 0 9 8 4 2 5 2 9 9
9 0 7 0 0 4 6 7 5 1 8 3 9 0 9 1 4 5 0 7 8 2 3 2 5 6 0 6 0 6 5 2 7 2 9 4 5 0
5 8 3 6 8 6 6 6 1 5 7 1 8 2 9 8 5 7 9 4 7 8 1 9 8 9 2 7 0 5 2 2 1 9 2 6 5 9
3 7 1 0 2 0 3 8 6 7 1 4 6 7 0 2 6 4 9 9 7 2 1 4 0 1 0 5 1 9 9 5 8 3 3 3 9 3
1 4 1 4 1 0 3 7 2 0 7 0 4 1 6 6 9 7 1 2 5 0 3 2 8 3 0 7 8 7 5 2 7 0 2 8 5 5
1 5 3 6 3 1 4 5 9 1 2 2 1 5 9 4 6 6 3 6 7 8 7 8 8 9 6 6 0 0 6 9 3 2 4 1 4 9
4 9 6 7 1 1 4 7 1 0 4 2 3 8 9 7 3 0 3 9 9 4 4 1 5 9 5 0 7 3 8 3 6 9 3 5 5 7
0 3 5 5 6 0 3 7 8 2 3 4 1 0 3 8 7 7 9 7 7 3 0 0 2 6 6 3 0 8 5 9 1 9 0 6 7 6

"Monte Carlo" simulations are typically used to randomly select values for input parameters based cumulative frequency distributions. A number between 0 and 1 is selected from a random number table (or program). This random number is converted into an input value using cumulative frequency distribution. Finally, the distribution of DCF-ROR's is calculated by repeating the method for all input variables.

Consider a mine with 40 operating units and 100 sales contracts. The units averaged the following approximate values over the last year:

- O&M Costs = $30/ton
- Market Price = $50/ton
- Saleable Production = 325 tons/hr

The site generates about $6,500/hr in profit (325 t/hr x $20/ton = $6,500/hr). The mine needs >$5,500/hr for new capital investments.

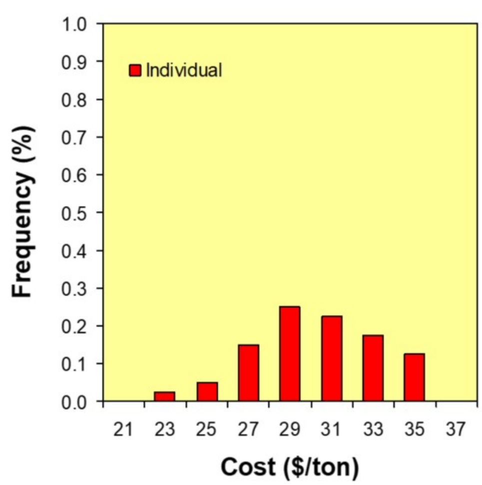
Would you recommend this level of investment to management?

First, let's look at the distribution of O&M costs associated with this site. O&M costs ranged from a low of $23.60/ton to a high of $35.90/ton. Nine cost categories were created starting at $20/ton with a $2/ton increment. The number of mining units in each increment was assigned based on costs. Individual and cumulative distributions were computed based on these values.

| Cost ($/ton) | No. Units |
|---|---|
| 20-22 | 0 |
| 22-24 | 1 |
| 24-26 | 2 |
| 26-28 | 6 |
| 28-30 | 10 |
| 30-32 | 9 |
| 32-34 | 7 |
| 34-36 | 5 |
| 36-38 | 0 |
| Total | 40 |

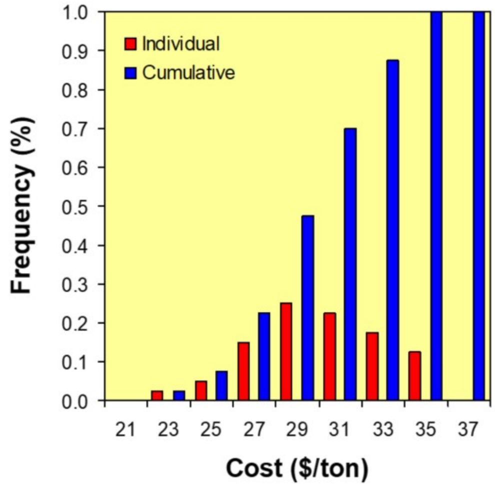
**O&M Cost Distribution**

| Cost ($/ton) | No. Units | Ind. (%) | Cum. (%) |
|---|---|---|---|
| 21 | 0 | 0.000 | 0.000 |
| 23 | 1 | 0.025 | 0.025 |
| 25 | 2 | 0.050 | 0.075 |
| 27 | 6 | 0.150 | 0.225 |
| 29 | 10 | 0.250 | 0.475 |
| 31 | 9 | 0.225 | 0.700 |
| 33 | 7 | 0.175 | 0.875 |
| 35 | 5 | 0.125 | 1.000 |
| 37 | 0 | 0.000 | 1.000 |
| Totals | 40 | 1.000 | |



## O&M Cost Distribution

| Cost ($/ton) | No. Units | Ind. (%) | Cum. (%) |
|---|---|---|---|
| 21 | 0 | 0.000 | 0.000 |
| 23 | 1 | 0.025 | 0.025 |
| 25 | 2 | 0.050 | 0.075 |
| 27 | 6 | 0.150 | 0.225 |
| 29 | 10 | 0.250 | 0.475 |
| 31 | 9 | 0.225 | 0.700 |
| 33 | 7 | 0.175 | 0.875 |
| 35 | 5 | 0.125 | 1.000 |
| 37 | 0 | 0.000 | 1.000 |
| Totals | 40 | 1.000 | |




Similar frequency distributions can also be created for other input variables.

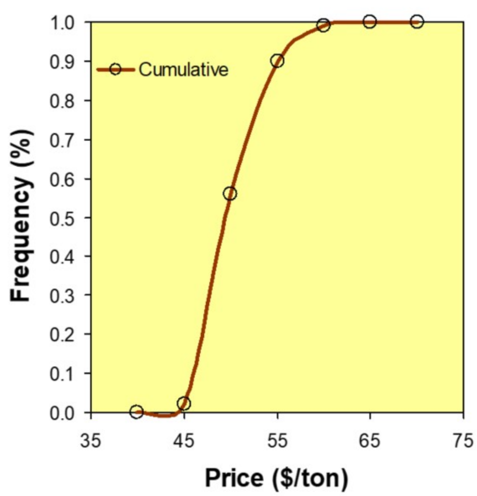
Let's add values for market price and mine production to the analysis.

| Price ($/ton) | No. Sales |
|---|---|
| <40 | 0 |
| 40-45 | 2 |
| 45-50 | 54 |
| 50-55 | 34 |
| 55-60 | 9 |
| 60-65 | 1 |
| 65-70 | 0 |
| 70-75 | 0 |
| >75 | 0 |
| Total | 100 |

| Prod. (tph) | No. Mines |
|---|---|
| <260 | 0 |
| 260-280 | 1 |
| 280-300 | 2 |
| 300-320 | 11 |
| 320-340 | 18 |
| 340-360 | 7 |
| 360-380 | 1 |
| 380-400 | 0 |
| >400 | 0 |
| Total | 40 |

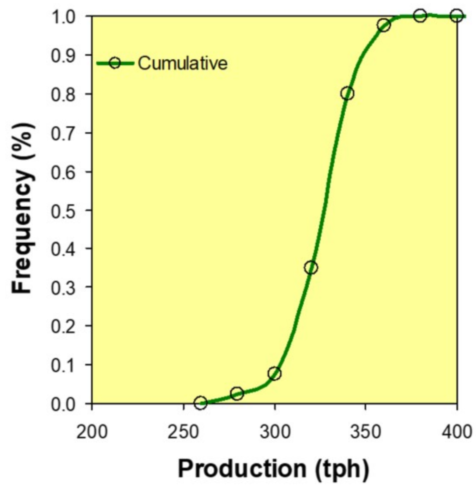
## Market Price Distribution

| Price ($/ton) | No. Sales | Ind. (%) | Cum. (%) |
|---|---|---|---|
| 37.5 | 0 | 0.000 | 0.000 |
| 42.5 | 2 | 0.020 | 0.020 |
| 47.5 | 54 | 0.540 | 0.560 |
| 52.5 | 34 | 0.340 | 0.900 |
| 57.5 | 9 | 0.090 | 0.990 |
| 62.5 | 1 | 0.010 | 1.000 |
| 67.5 | 0 | 0.000 | 1.000 |
| Totals | 100 | 1.000 | |



## Saleable Production Distribution

| Prod. (tph) | No. Units | Ind. (%) | Cum. (%) |
|---|---|---|---|
| 250 | 0 | 0.000 | 0.000 |
| 270 | 1 | 0.025 | 0.025 |
| 290 | 2 | 0.050 | 0.075 |
| 310 | 11 | 0.275 | 0.350 |
| 330 | 18 | 0.450 | 0.800 |
| 350 | 7 | 0.175 | 0.975 |
| 370 | 1 | 0.025 | 1.000 |
| 390 | 0 | 0.000 | 1.000 |
| 410 | 0 | 0.000 | 1.000 |
| Totals | 40 | 1.000 | |




Cumulative distributions are now available for:

- O&M Cost ($/ton)
- Market Price ($/ton)
- Saleable Production (tph)

Three random numbers between 0.000 and 1.000 need to be selected. Since three digits of precision is desired, nine random numbers have to be obtained (i.e., 3 digits/number x 3 numbers = 9 digits). The values can be manually selected from a random number table or generated by a software program (e.g., Excel).

## Random Number Table

| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 4 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | 4 | 3 | 0 | 7 | 4 | 2 | 2 | 6 | 8 | 2 | 6 | 6 | 2 | 2 | 1 | 4 | 8 | 6 | 3 | 9 | 1 | 3 | 8 | 4 | 1 | 4 | 1 | 0 | 8 |
| [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | 8 | 9 | 9 | 5 | 0 | 4 | 8 | 5 | 5 | 9 | 3 | 4 | 9 | 8 | 2 | 1 | 4 | 2 | 6 | 2 | 2 | 1 | 9 | 7 | 3 | 2 | 4 | 0 | 2 |
| [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | 3 | 4 | 2 | 6 | 4 | 1 | 3 | 0 | 1 | 8 | 4 | 5 | 6 | 2 | 8 | 4 | 4 | 3 | 8 | 9 | 3 | 0 | 2 | 2 | 6 | 6 | 2 | 3 | 9 |
| [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | 7 | 9 | 0 | 6 | 5 | 5 | 9 | 7 | 2 | 5 | 3 | 7 | 3 | 0 | 8 | 9 | 7 | 0 | 0 | 2 | 7 | 5 | 2 | 8 | 3 | 1 | 0 | 1 | 9 |
| [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] | 0 | 4 | 5 | 2 | 0 | 5 | 6 | 7 | 2 | 2 | 4 | 0 | 7 | 9 | 2 | 0 | 4 | 1 | 7 | 1 | 5 | 7 | 6 | 6 | 8 | 1 | 9 | 4 | 1 |
| 2 | 5 | 7 | 8 | 9 | 1 | [illegible] | [illegible] | 1 | 9 | 9 | 0 | 5 | 7 | 0 | 0 | 8 | 8 | 3 | 9 | 9 | 1 | 9 | 0 | 4 | 3 | 6 | 1 | 2 | 7 | 9 | 6 | 7 | 1 | 3 | 7 | 4 | 7 |
| 9 | 1 | 1 | 4 | 9 | 7 | 6 | [illegible] | [illegible] | 0 | 6 | 3 | 9 | 3 | 1 | 9 | 9 | 9 | 4 | 7 | 0 | 5 | 7 | 8 | 6 | 7 | 8 | 0 | 7 | 3 | 4 | 5 | 4 | 0 | 3 | 7 | 0 | 9 |
| 5 | 6 | 7 | 4 | 0 | 3 | 5 | 1 | 4 | 5 | 0 | 3 | 3 | 3 | 3 | 0 | 6 | 3 | 6 | 9 | 0 | 1 | 3 | 2 | 6 | 4 | 0 | 0 | 5 | 3 | 2 | 8 | 9 | 8 | 9 | 6 | 7 | 3 |
| 8 | 5 | 3 | 0 | 6 | 6 | 7 | 6 | 1 | 5 | 9 | 6 | 4 | 3 | 9 | 7 | 3 | 5 | 6 | 2 | 4 | 9 | 7 | 7 | 0 | 9 | 0 | 7 | 9 | 2 | 3 | 8 | 9 | 9 | 9 | 6 | 9 | 4 |
| 3 | 9 | 2 | 8 | 1 | 9 | 1 | 7 | 5 | 2 | 6 | 8 | 1 | 0 | 2 | 5 | 6 | 4 | 2 | 9 | 1 | 9 | 2 | 3 | 0 | 2 | 0 | 6 | 8 | 8 | 0 | 0 | 7 | 5 | 0 | 4 | 7 | 5 |
| 7 | 9 | 2 | 8 | 9 | 2 | 9 | 1 | 0 | 9 | 9 | 1 | 0 | 5 | 0 | 5 | 9 | 0 | 6 | 4 | 8 | 2 | 6 | 9 | 7 | 9 | 1 | 8 | 2 | 9 | 7 | 0 | 4 | 9 | 7 | 3 | 3 | 6 |
| 1 | 7 | 0 | 5 | 7 | 7 | 6 | 6 | 4 | 1 | 0 | 7 | 6 | 5 | 1 | 2 | 6 | 0 | 7 | 1 | 7 | 7 | 8 | 6 | 2 | 0 | 3 | 8 | 4 | 0 | 9 | 8 | 4 | 2 | 5 | 2 | 9 | 9 |
| 9 | 0 | 7 | 0 | 0 | 4 | 6 | 7 | 5 | 1 | 8 | 3 | 9 | 0 | 9 | 1 | 4 | 5 | 0 | 7 | 8 | 2 | 3 | 2 | 5 | 6 | 0 | 6 | 0 | 6 | 5 | 2 | 7 | 2 | 9 | 4 | 5 | 0 |
| 5 | 8 | 3 | 6 | 8 | 6 | 6 | 6 | 1 | 5 | 7 | 1 | 8 | 2 | 9 | 8 | 5 | 7 | 9 | 4 | 7 | 8 | 1 | 9 | 8 | 9 | 2 | 7 | 0 | 5 | 2 | 2 | 1 | 9 | 2 | 6 | 5 | 9 |
| 3 | 7 | 1 | 0 | 2 | 0 | 3 | 8 | 6 | 7 | 1 | 4 | 6 | 7 | 0 | 2 | 6 | 4 | 9 | 9 | 7 | 2 | 1 | 4 | 0 | 1 | 0 | 5 | 1 | 9 | 9 | 5 | 8 | 3 | 3 | 3 | 9 | 3 |
| 1 | 4 | 1 | 4 | 1 | 0 | 3 | 7 | 2 | 0 | 7 | 0 | 4 | 1 | 6 | 6 | 9 | 7 | 1 | 2 | 5 | 0 | 3 | 2 | 8 | 3 | 0 | 7 | 8 | 7 | 5 | 2 | 7 | 0 | 2 | 8 | 5 | 5 |
| 1 | 5 | 3 | 6 | 3 | 1 | 4 | 5 | 9 | 1 | 2 | 2 | 1 | 5 | 9 | 4 | 6 | 6 | 3 | 6 | 7 | 8 | 7 | 8 | 8 | 9 | 6 | 6 | 0 | 0 | 6 | 9 | 3 | 2 | 4 | 1 | 4 | 9 |
| 4 | 9 | 6 | 7 | 1 | 1 | 4 | 7 | 1 | 0 | 4 | 2 | 3 | 8 | 9 | 7 | 3 | 0 | 3 | 9 | 9 | 4 | 4 | 1 | 5 | 9 | 5 | 0 | 7 | 3 | 8 | 3 | 6 | 9 | 3 | 5 | 5 | 7 |
| 0 | 3 | 5 | 5 | 6 | 0 | 3 | 7 | 8 | 2 | 3 | 4 | 1 | 0 | 3 | 8 | 7 | 7 | 9 | 7 | 7 | 3 | 0 | 0 | 2 | 6 | 6 | 3 | 0 | 8 | 5 | 9 | 1 | 9 | 0 | 6 | 7 | 6 |

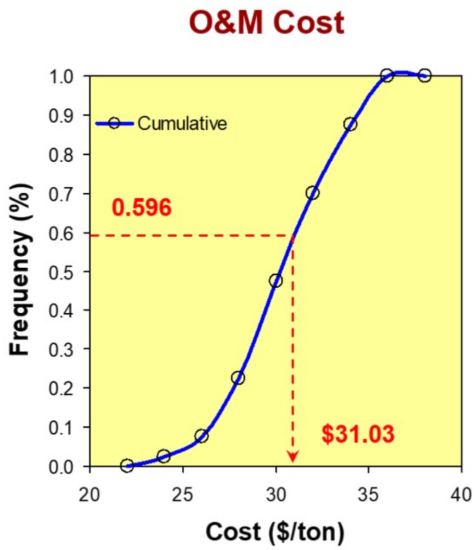
Choose an arbitrary starting point to select a group of numbers.

Random numbers …

- 596 →0.596→$31.03
- 439
- 735

Resultant value …

- Cost = $31.03/ton

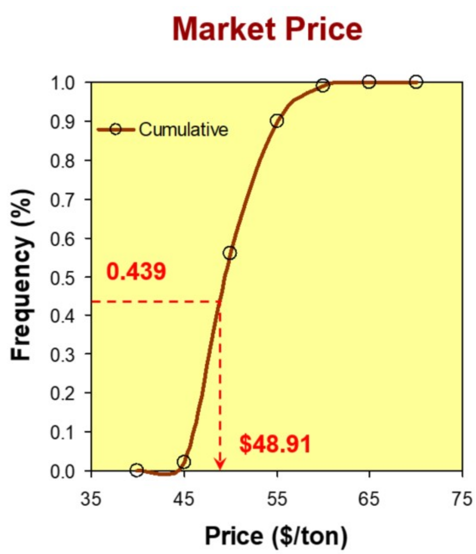



Random numbers …

- 596 →0.596→$31.03
- 439 →0.439→$48.91
- 735

Resultant value …

- Cost = $31.03/ton
- Price = $48.91/ton

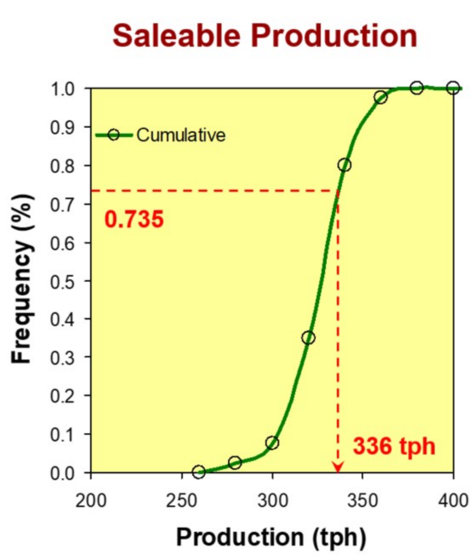



Random numbers …

- 596 →0.596→$31.03
- 439 →0.439→$48.91
- 735 →0.735→336 tph

Resultant value …

- Cost = $31.03/ton
- Price = $48.91/ton
- Prod. = 336 tph




Input values …

- Cost = $31.03/ton
- Price = $48.91/ton

- = 336 tph

Profit calculation …

Profit = Production {t/hr} x (Price-Cost) {$/ton}

Profit = 336 t/hr x ($48.91 - $31.03)/ton = $6,011/hr

The same simulation procedure was repeated 20 times to generate the following values:

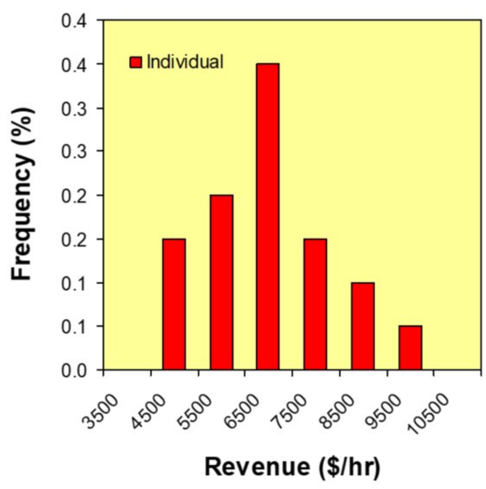
- $6,011 $5,914 $6,356 $5,111
- $4,831 $6,792 $6,597 $7,339
- $7,649 $4,945 $5,211 $6,040
- $7,567 $6,003 $9,634 $6,437
- $4,211 $5,981 $8,205 $8,202

From these values, we find:

- Mean = $6,452
- Standard Deviation = $1,342
- Max = $9,634; Min = $4,211

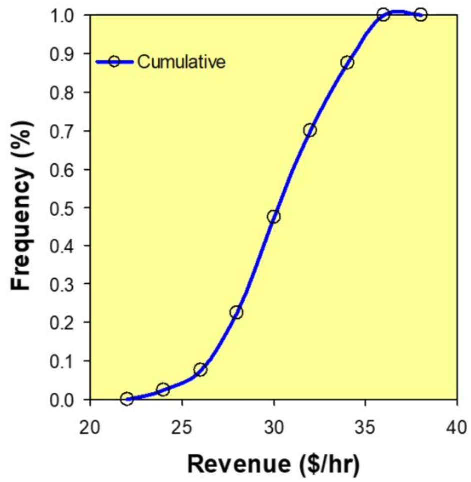
**Profit Distribution**

| Profit ($/hr) | No. Events | Ind. (%) | Cum. (%) |
|---|---|---|---|
| 3500 | 0 | 0.00 | 0.00 |
| 4500 | 3 | 0.15 | 0.15 |
| 5500 | 4 | 0.20 | 0.35 |
| 6500 | 7 | 0.35 | 0.70 |
| 7500 | 3 | 0.15 | 0.85 |
| 8500 | 2 | 0.10 | 0.95 |
| 9500 | 1 | 0.05 | 1.00 |
| 10500 | 0 | 0.00 | 1.00 |
| Totals | 20 | 1.000 | |



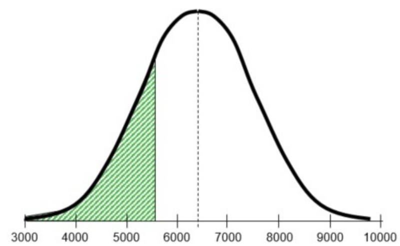
## Revenue Distribution

| Profit ($/hr) | No. Events | Ind. (%) | Cum. (%) |
|---|---|---|---|
| 3500 | 0 | 0.00 | 0.00 |
| 4500 | 3 | 0.15 | 0.15 |
| 5500 | 4 | 0.20 | 0.35 |
| 6500 | 7 | 0.35 | 0.70 |
| 7500 | 3 | 0.15 | 0.85 |
| 8500 | 2 | 0.10 | 0.95 |
| 9500 | 1 | 0.05 | 1.00 |
| 10500 | 0 | 0.00 | 1.00 |
| Totals | 20 | 1.000 | |

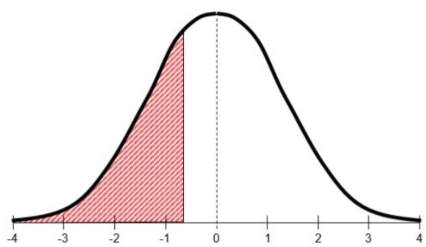



Example

- Assume profit is normally distributed.
  - μ=$6,452
  - σ=$1,342.
- Find the probability that P{X<5,500}.
  - Z=(5500-6452)/1342
    = -0.709
  - P{X<5500} = P{Z<-0.709}.




$$P\left\{\frac{X-\mu}{\sigma} = \frac{5500-6452}{1342}\right\}$$



## Example

- Find the probability that P{X<5,500}.
  - Z=(5500-6452)/1342 = -0.709
  - P{X<5500} = P{Z<-0.709}
  - P{Z<-0.709} = 1-0.7852 = 0.2248
- Only a 22% chance profit expectations will not be met.

**Z-STATISTIC PROBABILITIES (AREA UNDER CURVE)**

| Z | 0.00 | 0.01 | 0.02 | 0.03 | 0.04 | 0.05 | 0.06 | 0.07 | 0.08 | 0.09 |
|---|---|---|---|---|---|---|---|---|---|---|
| 0.0 | 0.5000 | 0.5040 | 0.5080 | 0.5120 | 0.5160 | 0.5199 | 0.5239 | 0.5279 | 0.5319 | 0.5359 |
| 0.1 | 0.5398 | 0.5438 | 0.5478 | 0.5517 | 0.5557 | 0.5596 | 0.5636 | 0.5675 | 0.5714 | 0.5753 |
| 0.2 | 0.5793 | 0.5832 | 0.5871 | 0.5910 | 0.5948 | 0.5987 | 0.6026 | 0.6064 | 0.6103 | 0.6141 |
| 0.3 | 0.6179 | 0.6217 | 0.6255 | 0.6293 | 0.6331 | 0.6368 | 0.6406 | 0.6443 | 0.6480 | 0.6517 |
| 0.4 | 0.6554 | 0.6591 | 0.6628 | 0.6664 | 0.6700 | 0.6736 | 0.6772 | 0.6808 | 0.6844 | 0.6879 |
| 0.5 | 0.6915 | 0.6950 | 0.6985 | 0.7019 | 0.7054 | 0.7088 | 0.7123 | 0.7157 | 0.7190 | 0.7224 |
| 0.6 | 0.7257 | 0.7291 | 0.7324 | 0.7357 | 0.7389 | 0.7422 | 0.7454 | 0.7486 | 0.7517 | 0.7549 |
| 0.7 | 0.7580 | 0.7611 | 0.7642 | 0.7673 | 0.7704 | 0.7734 | 0.7764 | 0.7794 | 0.7823 | 0.7852 |
| 0.8 | 0.7881 | 0.7910 | 0.7939 | 0.7967 | 0.7995 | 0.8023 | 0.8051 | 0.8078 | 0.8106 | 0.8133 |
| 0.9 | 0.8159 | 0.8186 | 0.8212 | 0.8238 | 0.8264 | 0.8289 | 0.8315 | 0.8340 | 0.8365 | 0.8389 |
| 1.0 | 0.8413 | 0.8438 | 0.8461 | 0.8485 | 0.8508 | 0.8531 | 0.8554 | 0.8577 | 0.8599 | 0.8621 |
| 1.1 | 0.8643 | 0.8665 | 0.8686 | 0.8708 | 0.8729 | 0.8749 | 0.8770 | 0.8790 | 0.8810 | 0.8830 |
| 1.2 | 0.8849 | 0.8869 | 0.8888 | 0.8907 | 0.8925 | 0.8944 | 0.8962 | 0.8980 | 0.8997 | 0.9015 |
| 1.3 | 0.9032 | 0.9049 | 0.9066 | 0.9082 | 0.9099 | 0.9115 | 0.9131 | 0.9147 | 0.9162 | 0.9177 |
| 1.4 | 0.9192 | 0.9207 | 0.9222 | 0.9236 | 0.9251 | 0.9265 | 0.9279 | 0.9292 | 0.9306 | 0.9319 |
| 1.5 | 0.9332 | 0.9345 | 0.9357 | 0.9370 | 0.9382 | 0.9394 | 0.9406 | 0.9418 | 0.9429 | 0.9441 |
| 1.6 | 0.9452 | 0.9463 | 0.9474 | 0.9484 | 0.9495 | 0.9505 | 0.9515 | 0.9525 | 0.9535 | 0.9545 |
| 1.7 | 0.9554 | 0.9564 | 0.9573 | 0.9582 | 0.9591 | 0.9599 | 0.9608 | 0.9616 | 0.9625 | 0.9633 |
| 1.8 | 0.9641 | 0.9649 | 0.9656 | 0.9664 | 0.9671 | 0.9678 | 0.9686 | 0.9693 | 0.9699 | 0.9706 |
| 1.9 | 0.9713 | 0.9719 | 0.9726 | 0.9732 | 0.9738 | 0.9744 | 0.9750 | 0.9756 | 0.9761 | 0.9767 |
| 2.0 | 0.9772 | 0.9778 | 0.9783 | 0.9788 | 0.9793 | 0.9798 | 0.9803 | 0.9808 | 0.9812 | 0.9817 |
| 2.1 | 0.9821 | 0.9826 | 0.9830 | 0.9834 | 0.9838 | 0.9842 | 0.9846 | 0.9850 | 0.9854 | 0.9857 |
| 2.2 | 0.9861 | 0.9864 | 0.9868 | 0.9871 | 0.9875 | 0.9878 | 0.9881 | 0.9884 | 0.9887 | 0.9890 |
| 2.3 | 0.9893 | 0.9896 | 0.9898 | 0.9901 | 0.9904 | 0.9906 | 0.9909 | 0.9911 | 0.9913 | 0.9916 |
| 2.4 | 0.9918 | 0.9920 | 0.9922 | 0.9925 | 0.9927 | 0.9929 | 0.9931 | 0.9932 | 0.9934 | 0.9936 |
| 2.5 | 0.9938 | 0.9940 | 0.9941 | 0.9943 | 0.9945 | 0.9946 | 0.9948 | 0.9949 | 0.9951 | 0.9952 |
| 2.6 | 0.9953 | 0.9955 | 0.9956 | 0.9957 | 0.9959 | 0.9960 | 0.9961 | 0.9962 | 0.9963 | 0.9964 |
| 2.7 | 0.9965 | 0.9966 | 0.9967 | 0.9968 | 0.9969 | 0.9970 | 0.9971 | 0.9972 | 0.9973 | 0.9974 |
| 2.8 | 0.9974 | 0.9975 | 0.9976 | 0.9977 | 0.9977 | 0.9978 | 0.9979 | 0.9979 | 0.9980 | 0.9981 |
| 2.9 | 0.9981 | 0.9982 | 0.9982 | 0.9983 | 0.9984 | 0.9984 | 0.9985 | 0.9985 | 0.9986 | 0.9986 |
| 3.0 | 0.9987 | 0.9987 | 0.9987 | 0.9988 | 0.9988 | 0.9989 | 0.9989 | 0.9989 | 0.9990 | 0.9990 |
| 3.1 | 0.9990 | 0.9991 | 0.9991 | 0.9991 | 0.9992 | 0.9992 | 0.9992 | 0.9992 | 0.9993 | 0.9993 |
| 3.2 | 0.9993 | 0.9993 | 0.9994 | 0.9994 | 0.9994 | 0.9994 | 0.9994 | 0.9995 | 0.9995 | 0.9995 |
| 3.3 | 0.9995 | 0.9995 | 0.9995 | 0.9996 | 0.9996 | 0.9996 | 0.9996 | 0.9996 | 0.9996 | 0.9997 |
| 3.4 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9997 | 0.9998 |
| 3.5 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 | 0.9998 |

The “coefficient of variation” for this exercise suggests a relatively low risk.  Ultimately, a judgment call will have to be made by the operator based on these analyses.

# 16.0 Simulation Issues

Monte Carlo analyses assume that variables are independent of each other. Unfortunately, this is often not the case in real world problems. For example, in mining, price and cost are often related. Therefore, the “degree of correlation” may need to be included in the analysis.

The Correlation can be quantified using “Pearson’s Coefficient (r)” given by:

$$r = \frac{\sum X_i Y_i - (\sum X_i \sum Y_i)/N}{(N-1)(s_x s_y)}$$

N = number of data pairs

$X_i, Y_i$ = observation of X or Y

$s_x$, $s_y$ = standard deviation for X and Y

The r value indicates:

- No correlation (r=0)
- Strong positive correlation (r=1)
- Strong negative correlation (r=-1)

For example, a comparison of copper, lead and zinc prices gave the following.

| Price | Copper | Zinc | Lead |
|---|---|---|---|
| Copper | 1.00 | -- | -- |
| Zinc | 0.80 | 1.00 | -- |
| Lead | 0.72 | 0.89 | 1.00 |

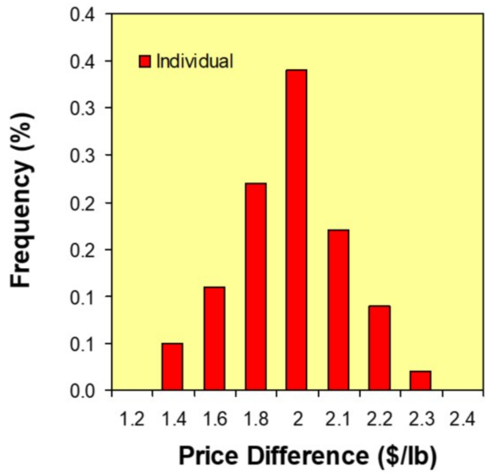
Thus, a price correlation does appear to exist for these metals.

Interdependence is included in Monte Carlo analysis using “delta distributions.” After choosing a primary independent variable, other variables can be related by difference (i.e., delta between variables). For example, if copper price was chosen as the independent variable, the other metal prices can be determined based on a difference in dollars per pound.

**Metal Price Distribution**

| Delta* ($/lb) | Ind. (%) | Cum. (%) |
|---|---|---|
| 1.20 | 0.00 | 0.00 |
| 1.40 | 0.05 | 0.05 |
| 1.60 | 0.11 | 0.16 |
| 1.80 | 0.22 | 0.38 |
| 2.00 | 0.34 | 0.72 |
| 2.10 | 0.17 | 0.89 |
| 2.20 | 0.09 | 0.98 |
| 2.30 | 0.02 | 1.00 |
| 2.40 | 0.00 | 1.00 |
| Totals | 1.000 | |

*Delta = ($/lb Cu) – ($/lb Pb)




Since metal prices are not perfectly correlated, random numbers must be selected again and used to determine the pricing differentials. This procedure can be used with almost any set of variables to construct a suitable Monte Carlo analysis with both independent and correlated variables. Ultimately, a distribution of DCF-ROR values can be established for assessing project risk using this approach.

# 17.0 Discounted Cash Flow - Rate of Return

Corporations often handle financial decisions using a Discounted Cash Flow – Rate of Return (DCF-ROR) analysis. DCF-ROR is the after-tax rate of return that properly discounts future cash flow. This is sometimes referred to as “internal”, “true” or “investor” rate of return. Polls indicate this is the decision method used by more than 90% of U.S. companies. DCF-ROR is considered better than other methods (e.g., payback period) since they fail to properly account for the tax implications of the time value of money.

DCF-ROR is the rate of return that makes the present worth of future generated cash flow over the life of a project (including after tax salvage value) equal to the present worth of all after-tax investments. DCF-ROR involves discounting future cash flow to a present value according to mathematics of compound interest at some interest rate. DCF-ROR is the determination of the interest rate that equates to the present worth of future after-tax earnings with the present worth of all after-tax investments necessary to create such earnings.

The basis for DCF-ROR is that today’s dollar is worth more than tomorrow’s dollar since it can be invested to earn money in the interim. Future dollars in cash flow schedules are therefore “discounted”. The higher the “discount rate”, the less the future dollar is worth today. This concept is applicable to all capital projects regardless of the dollar value. It provides effective and consistent evaluation of investment opportunities and determines the most financially attractive projects (critical to decision making). However, its results are heavily dependent upon the validity and reliability of assumptions/predictions.

The process of DCF-ROR is as follows:

Step 1 – Calculate Annual Cash Flow

- Determine cash flow by accounting for gross profit, book deductions and taxes.

Step 2 - Construct cash flow diagram

- Draw a timeline of all cash flow values.
- Assures all after-tax costs, cash flows and salvages are properly related in time.

Step 3 - Formulate present worth equation

- Convert data in cash flow diagram into a mathematical expression of present worth using compound interest formula.
- Equation must “discount” future values.

Step 4 - Solve for rate of return

- Use the trial-and-error (or iterative) approach to solve for the rate of return that makes the present worth equal to zero.

- For larger problems, it is usually best to prepare a spreadsheet and solve via the “Goal Seek” or “Solver” tools.

Step 5 – Accept/reject project investment

- Compare solution with minimum acceptable rate of return (MARR).
- May also consider other factors (e.g., reputation, environmental impact, etc.).

For a corporation, “cash flow” is defined as after tax earnings available for use after meeting all expenses (including taxes).

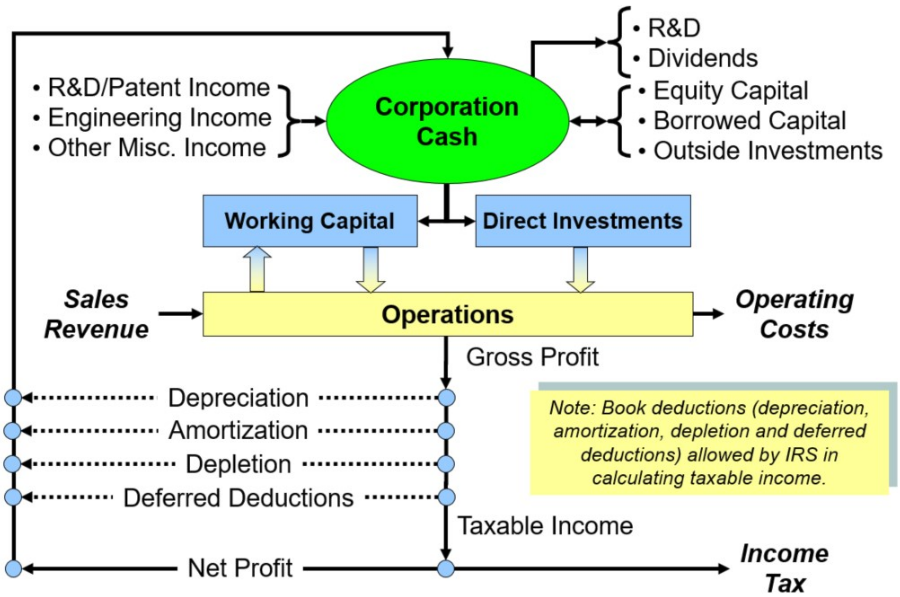
Cash flow is equivalent to:

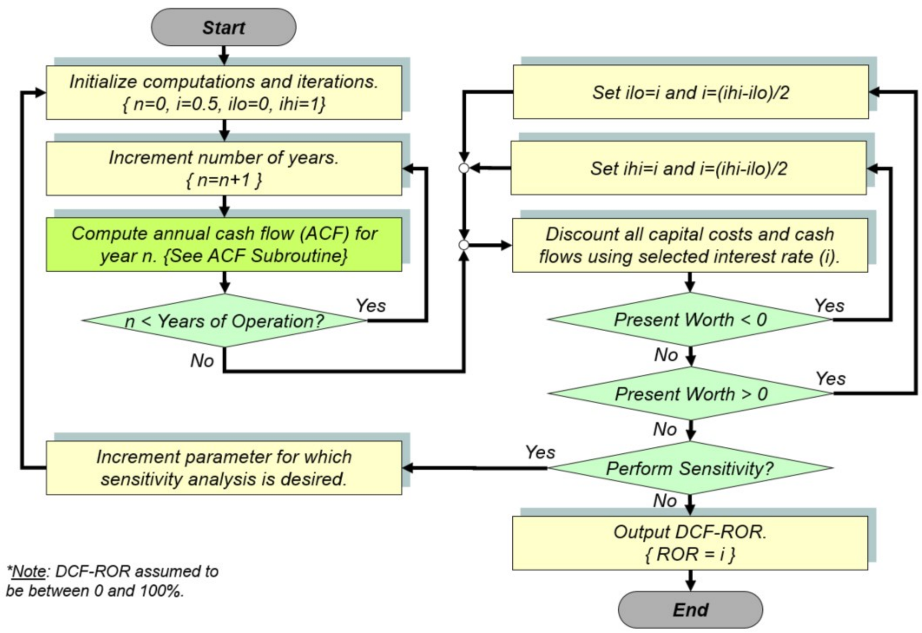
- The sum of net profit, depreciation, amortization, depletion and deferred deduction items, or
- sales revenue minus operating costs and income taxes.

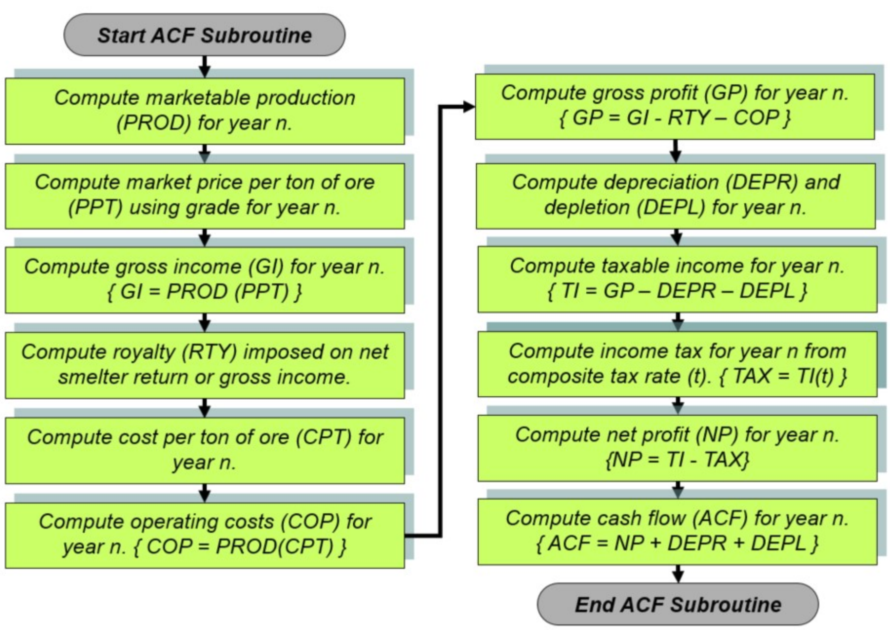
Pre-tax “book deductions” are allowed so that investors can recover capital expenditures in a business.

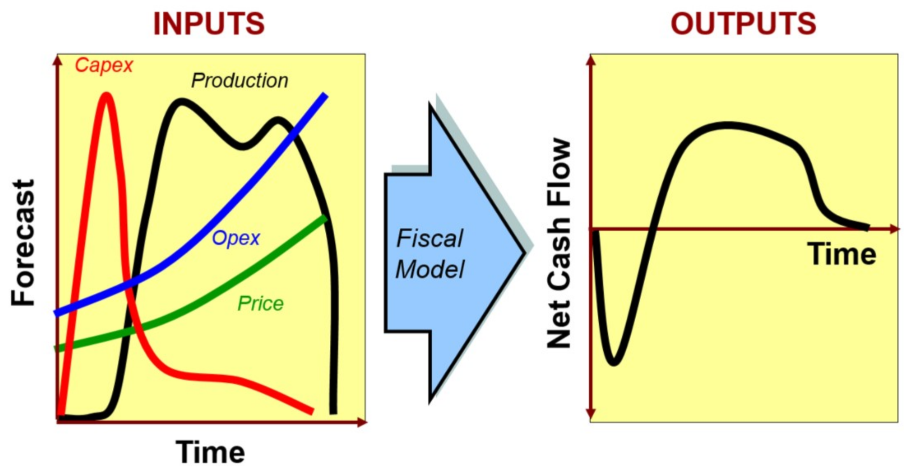
- Just a deduction – not an actual money disbursement (IRS is not sending you money).
- Available funds encourage reinvestment.

| | |
|---|---|
| **Gross Income** | = grade x recovery x price x tonnage |
| - Royalty | = imposed on either gross income or gross profits |
| - Operating Costs | = mining + processing + transport + etc. |
| **Gross Profit** | = sales revenue minus operating costs and royalty |
| - Depreciation | |
| - Depletion | |
| - Deferred Deductions | |
| **Taxable Income** | |
| - Income Tax | = national x federal rate + local x local rate |
| **Net Profit** | |
| + Depreciation | = depreciation, amortization, depletion and deferred deductions are “book deductions” allowed by IRS |
| + Depletion | |
| + Deferred Deductions | |
| **Annual Cash Flow** | = value placed into cash flow diagram |


• R&D
• Dividends
• R&D/Patent Income
• Engineering Income
• Other Misc. Income
Corporation Cash
• Equity Capital
• Borrowed Capital
• Outside Investments
Working Capital
Direct Investments
Sales Revenue
Operations
Operating Costs
Gross Profit
Depreciation
Amortization
Depletion
Deferred Deductions
Note: Book deductions (depreciation, amortization, depletion and deferred deductions) allowed by IRS in calculating taxable income.
Taxable Income
Net Profit
Income Tax


Start
Initialize computations and iterations. { n=0, i=0.5, ilo=0, ihi=1}
Increment number of years. { n=n+1 }
Compute annual cash flow (ACF) for year n. {See ACF Subroutine}
n < Years of Operation?
Yes
No
Set ilo=i and i=(ihi-ilo)/2
Set ihi=i and i=(ihi-ilo)/2
Discount all capital costs and cash flows using selected interest rate (i).
Present Worth < 0
Yes
No
Present Worth > 0
Yes
No
Perform Sensitivity?
Yes
No
Increment parameter for which sensitivity analysis is desired.
Output DCF-ROR. { ROR = i }
End
*Note: DCF-ROR assumed to be between 0 and 100%.


Start ACF Subroutine
Compute marketable production (PROD) for year n.
Compute market price per ton of ore (PPT) using grade for year n.
Compute gross income (GI) for year n. { GI = PROD (PPT) }
Compute royalty (RTY) imposed on net smelter return or gross income.
Compute cost per ton of ore (CPT) for year n.
Compute operating costs (COP) for year n. { COP = PROD(CPT) }
Compute gross profit (GP) for year n. { GP = GI - RTY – COP }
Compute depreciation (DEPR) and depletion (DEPL) for year n.
Compute taxable income for year n. { TI = GP – DEPR – DEPL }
Compute income tax for year n from composite tax rate (t). { TAX = TI(t) }
Compute net profit (NP) for year n. {NP = TI - TAX}
Compute cash flow (ACF) for year n. { ACF = NP + DEPR + DEPL }
End ACF Subroutine




Input Elements

Capex (Capital Cost Estimate)

- Expenditures are required to obtain the forecast benefits of a project.
- Includes the acquisition of property, construction, equipment, development costs, etc.

Opex (Operating Cost Estimate)

- Fixed – Operating costs are directly attributable to project, but unrelated to a level of activity (e.g., manpower, maintenance, etc.).
- Variable – Operating costs are directly attributable to level of activity (e.g., fuel, power, etc.)
- Overhead – Operating costs associated with administrative functions (e.g., accounting, R&D, etc.).

Production Forecast

- Estimate of marketable production that results in the generation of revenue.
- In mining, this may include elements such as recoverable reserves, mining plan/schedule, equipment performance, manpower skills, etc.

Price Forecast

- Estimate of the likely market price to be realized for sales of production units.
- Inaccurate predictions are common in volatile commodity markets.
- Forecasts are usually based on long and short term internal and external (consultant) projections.

Figures 17.1 – 17.3 show commodity price graphs for coal, uranium, and gold, respectively.

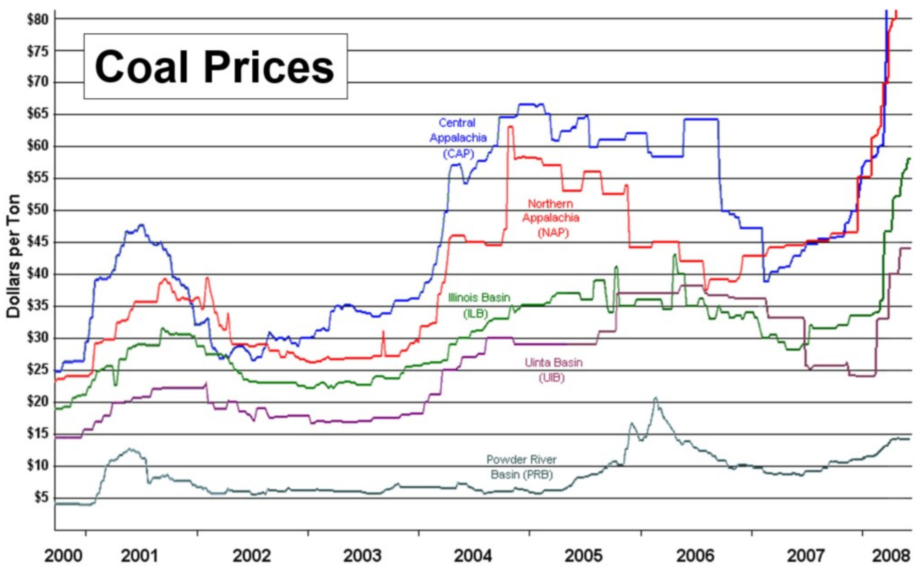



**Figure 17.1: Graph of Coal prices per ton.**

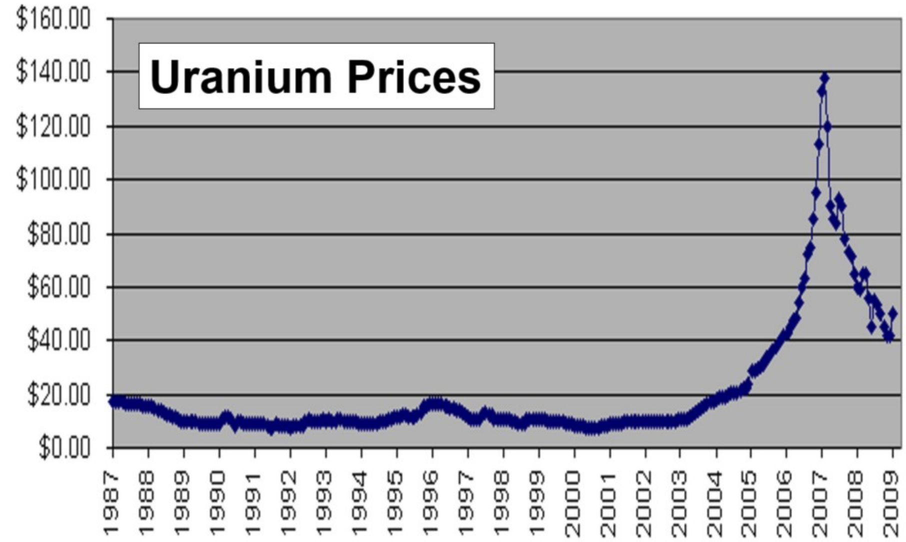



**Figure 17.2: Graph of Uranium prices per lb.**

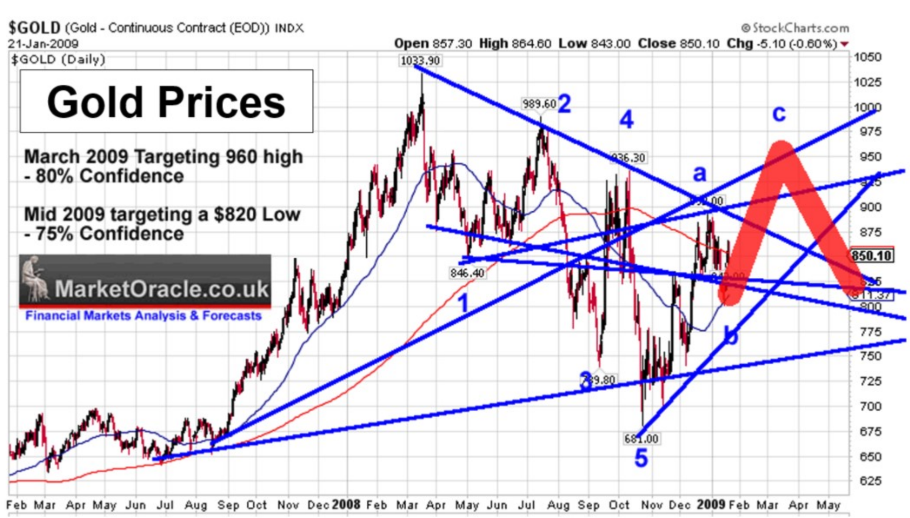


**Figure 17.3: Graph of Gold prices per oz.**

Some investment costs may be deducted or "expensed" in the year they occur. Development costs often fall in this category. There is a need for sufficient taxable income in the year to offset "expensed" costs. This is often necessary for tax credits since they only apply in the year the investment is made. If there is insufficient taxable income in that year, then deductions must be carried forward and used against project earnings as taxable income is generated in the future.

"Working capital" is the money required for the day-to-day operation of a business. This represents "what you can get your hands on" versus "what is tied up in assets". Working capital is NOT an allowable tax deduction (a very important realization). Working capital cannot be depreciated, depleted, amortized or expensed. It is represented in a CFD as an initial negative cost that is fully recovered at the project end and treated much like a 100% salvage value. In actuality, it may be recovered at some point in time or never fully recovered.

Example

| Period (Year End) | 0 | 1 | 2 |
|---|---|---|---|
| O&M Cost ($/ton) | $ 50.00 | $ 50.00 | $ 50.00 |
| Royalty Rate (NSR, %) | 10 | 10 | 10 |
| Depletion Rate | 22 | 22 | 22 |
| Tax Rate | 50 | 50 | 50 |
| Production (ton/yr) | **** | 100,000 | 100,000 |
| Grade (oz/ton) | **** | 5 | 8 |
| Recovery (%) | **** | 85 | 85 |
| Metal Price ($/oz) | **** | $ 20.00 | $ 20.00 |
| Sales Revenue ($/yr) | **** | $ 8,500,000 | $ 13,600,000 |
| O&M Cost ($/yr) | **** | $ 5,000,000 | $ 5,000,000 |
| Other Income ($/yr) | **** | $ - | $ - |
| Other Costs ($/yr) | **** | $ - | $ - |
| Gross Income ($/yr) | **** | $ 8,500,000 | $ 13,600,000 |
| - Operating Costs | **** | $ 5,000,000 | $ 5,000,000 |
| - Royalty | **** | $ 850,000 | $ 1,360,000 |
| Gross Profit/Loss ($/yr) | **** | $ 2,650,000 | $ 7,240,000 |
| - Depreciation | **** | $ 2,143,500 | $ 3,673,500 |
| - Depletion | **** | $ 253,250 | $ 1,783,250 |
| - Tax Loss Forward | **** | **** | $ - |
| Taxable Income ($/yr) | **** | $ 253,250 | $ 1,783,250 |
| - Income Tax | **** | $ 126,625 | $ 891,625 |
| + Tax Credit | **** | $ - | $ - |
| Net Profit/Loss ($/yr) | **** | $ 126,625 | $ 891,625 |
| + Depreciation | **** | $ 2,143,500 | $ 3,673,500 |
| + Depletion | **** | $ 253,250 | $ 1,783,250 |
| Cash Flow ($/yr) | $ - | $ 2,523,375 | $ 6,348,375 |
| - Capitalized Cost | $ 15,000,000 | | $ - |
| - Working Capital | $ 1,000,000 | $ - | $ - |
| Net Cash Flow ($/yr) | $ (16,000,000) | $ 2,523,375 | $ 6,348,375 |

| | |
|---|---|
| Carry Tax Loss Forward? | Yes |
| Discount Rate (%) | 15 |
| Net Present Value | $ 18,738,950 |
| Net Present Value=0 | ($0.00) |
| Discount Rate (%) | 37.01 |

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| Period (Year End) | 0 | 1 | 2 |
|---|---|---|---|
| O&M Cost ($/ton) | $ 50.00 | $ 50.00 | $ 50.00 |
| Royalty Rate (NSR, %) | 10 | 10 | 10 |
| Depletion Rate | 22 | 22 | 22 |
| Tax Rate | 50 | 50 | 50 |
| Production (ton/yr) | **** | 100,000 | 100,000 |
| Grade (oz/ton) | **** | 5 | 8 |
| Recovery (%) | **** | 85 | 85 |
| Metal Price ($/oz) | **** | $ 20.00 | $ 20.00 |
| Sales Revenue ($/yr) | **** | $ 8,500,000 | $ 13,600,000 |
| O&M Cost ($/yr) | **** | $ 5,000,000 | $ 5,000,000 |
| Other Income ($/yr) | **** | $ - | $ - |
| Other Costs ($/yr) | **** | $ - | $ - |
| Gross Income ($/yr) | **** | $ 8,500,000 | $ 13,600,000 |
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| Cash Flow ($/yr) | $ - | $ 2,523,375 | $ 6,348,375 |
| - Capitalized Cost | $ 15,000,000 | | $ - |
| - Working Capital | $ 1,000,000 | $ - | $ - |
| Net Cash Flow ($/yr) | $ (16,000,000) | $ 2,523,375 | $ 6,348,375 |

| | |
|---|---|
| Carry Tax Loss Forward? | Yes |
| Discount Rate (%) | 15 |
| Net Present Value | $ 18,738,950 |
| Net Present Value=0 | ($0.00) |
| Discount Rate (%) | 37.01 |

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| Period (Year End) | 0 | 1 | 2 |
|---|---|---|---|
| O&M Cost ($/ton) | $ 50.00 | $ 50.00 | $ 50.00 |
| Royalty Rate (NSR, %) | 10 | 10 | 10 |
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| Production (ton/yr) | **** | 100,000 | 100,000 |
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| O&M Cost ($/yr) | **** | $ 5,000,000 | $ 5,000,000 |
| Other Income ($/yr) | **** | $ - | $ - |
| Other Costs ($/yr) | **** | $ - | $ - |
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| - Tax Loss Forward | **** | **** | $ - |
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| + Depletion | **** | $ 253,250 | $ 1,783,250 |
| Cash Flow ($/yr) | $ - | $ 2,523,375 | $ 6,348,375 |
| - Capitalized Cost | $ 15,000,000 | | $ - |
| - Working Capital | $ 1,000,000 | $ - | $ - |
| Net Cash Flow ($/yr) | $ (16,000,000) | $ 2,523,375 | $ 6,348,375 |

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|---|---|
| Carry Tax Loss Forward? | Yes |
| Discount Rate (%) | 15 |
| Net Present Value | $ 18,738,950 |
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| Discount Rate (%) | 37.01 |

Next

| Period (Year End) | 0 | 1 | 2 |
|---|---|---|---|
| O&M Cost ($/ton) | $ 50.00 | $ 50.00 | $ 50.00 |
| Royalty Rate (NSR, %) | 10 | 10 | 10 |
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| O&M Cost ($/yr) | **** | $ 5,000,000 | $ 5,000,000 |
| Other Income ($/yr) | **** | $ - | $ - |
| Other Costs ($/yr) | **** | $ - | $ - |
| Gross Income ($/yr) | **** | $ 8,500,000 | $ 13,600,000 |
| - Operating Costs | **** | $ 5,000,000 | $ 5,000,000 |
| - Royalty | **** | $ 850,000 | $ 1,360,000 |
| Gross Profit/Loss ($/yr) | **** | $ 2,650,000 | $ 7,240,000 |
| - Depreciation | **** | $ 2,143,500 | $ 3,673,500 |
| - Depletion | **** | $ 253,250 | $ 1,783,250 |
| - Tax Loss Forward | **** | **** | $ - |
| Taxable Income ($/yr) | **** | $ 253,250 | $ 1,783,250 |
| - Income Tax | **** | $ 126,625 | $ 891,625 |
| + Tax Credit | **** | $ - | $ - |
| Net Profit/Loss ($/yr) | **** | $ 126,625 | $ 891,625 |
| + Depreciation | **** | $ 2,143,500 | $ 3,673,500 |
| + Depletion | **** | $ 253,250 | $ 1,783,250 |
| Cash Flow ($/yr) | $ - | $ 2,523,375 | $ 6,348,375 |
| - Capitalized Cost | $ 15,000,000 | | $ - |
| - Working Capital | $ 1,000,000 | $ - | $ - |
| Net Cash Flow ($/yr) | $ (16,000,000) | $ 2,523,375 | $ 6,348,375 |

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|---|---|
| Carry Tax Loss Forward? | Yes |
| Discount Rate (%) | 15 |
| Net Present Value | $ 18,738,950 |
| Net Present Value=0 | ($0.00) |
| Discount Rate (%) | 37.01 |

Next

| Period (Year End) | 0 | 1 | 2 |
|---|---|---|---|
| O&M Cost ($/ton) | $ 50.00 | $ 50.00 | $ 50.00 |
| Royalty Rate (NSR, %) | 10 | 10 | 10 |
| Depletion Rate | 22 | 22 | 22 |
| Tax Rate | 50 | 50 | 50 |
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|---|---|
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| Discount Rate (%) | 15 |
| Net Present Value | $ 18,738,950 |
| Net Present Value=0 | ($0.00) |
| Discount Rate (%) | 37.01 |

Next

| Period (Year End) | 0 | 1 | 2 |
|---|---|---|---|
| O&M Cost ($/ton) | $ 50.00 | $ 50.00 | $ 50.00 |
| Royalty Rate (NSR, %) | 10 | 10 | 10 |
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| Other Income ($/yr) | **** | $ - | $ - |
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| - Tax Loss Forward | **** | **** | $ - |
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| Cash Flow ($/yr) | $ - | $ 2,523,375 | $ 6,348,375 |
| - Capitalized Cost | $ 15,000,000 | | $ - |
| - Working Capital | $ 1,000,000 | $ - | $ - |
| Net Cash Flow ($/yr) | $ (16,000,000) | $ 2,523,375 | $ 6,348,375 |

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|---|---|
| Carry Tax Loss Forward? | Yes |
| Discount Rate (%) | 15 |
| Net Present Value | $ 18,738,950 |
| Net Present Value=0 | ($0.00) |
| Discount Rate (%) | 37.01 |

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| Period (Year End) | 0 | 1 | 2 |
|---|---|---|---|
| O&M Cost ($/ton) | $ 50.00 | $ 50.00 | $ 50.00 |
| Royalty Rate (NSR, %) | 10 | 10 | 10 |
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| Net Cash Flow ($/yr) | $ (16,000,000) | $ 2,523,375 | $ 6,348,375 |

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|---|---|
| Carry Tax Loss Forward? | Yes |
| Discount Rate (%) | 15 |
| Net Present Value | $ 18,738,950 |
| Net Present Value=0 | ($0.00) |
| Discount Rate (%) | 37.01 |

Next

| Period (Year End) | 0 | 1 | 2 |
|---|---|---|---|
| O&M Cost ($/ton) | $ 50.00 | $ 50.00 | $ 50.00 |
| Royalty Rate (NSR, %) | 10 | 10 | 10 |
| Depletion Rate | 22 | 22 | 22 |
| Tax Rate | 50 | 50 | 50 |
| Production (ton/yr) | **** | 100,000 | 100,000 |
| Grade (oz/ton) | **** | 5 | 8 |
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| Sales Revenue ($/yr) | **** | $ 8,500,000 | $ 13,600,000 |
| O&M Cost ($/yr) | **** | $ 5,000,000 | $ 5,000,000 |
| Other Income ($/yr) | **** | $ - | $ - |
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|---|---|
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| Net Present Value | $ 18,738,950 |
| Net Present Value=0 | ($0.00) |
| Discount Rate (%) | 37.01 |

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| Period (Year End) | 0 | 1 | 2 |
|---|---|---|---|
| O&M Cost ($/ton) | $ 50.00 | $ 50.00 | $ 50.00 |
| Royalty Rate (NSR, %) | 10 | 10 | 10 |
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| Tax Rate | 50 | 50 | 50 |
| Production (ton/yr) | **** | 100,000 | 100,000 |
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| Period (Year End) | 0 | 1 | 2 |
|---|---|---|---|
| O&M Cost ($/ton) | $ 50.00 | $ 50.00 | $ 50.00 |
| Royalty Rate (NSR, %) | 10 | 10 | 10 |
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|---|---|
| Carry Tax Loss Forward? | Yes |
| Discount Rate (%) | 15 |
| Net Present Value | $ 18,738,950 |
| Net Present Value=0 | ($0.00) |
| Discount Rate (%) | 37.01 |

What about other approaches?

While DCF-ROR uses "present worth" calculations by standard convention, identical conclusions can be obtained by comparing income and costs at any other fixed point in time (e.g., future worth). The challenge in DCF-ROR analysis to properly determine annual cash flows over the project life by accounting for all capital costs, O&M expenditures, revenues, and salvage values with appropriate tax considerations.

# Ventilation Surveys

Jump to Today | Edit

**Mine Ventilation Surveys**

This course is part of a larger suite of courses covering a variety of topics in mine ventilation. A survey or audit of the ventilation system can refer to a range of activities and purposes depending on the context. This course will focus on the technical aspects of planning measuring and analyzing ventilation system parameters. Additional, conceptual information regarding context and purpose may be found in the course modules for “Health Effects, Measurement and Control of Diesel Emissions”, “Metal / Nonmetal Ventilation System Design” and “Ventilation Network Simulation and Modeling” attached to this series.

**Learning Objectives**

1. **Demonstrate how to plan a ventilation survey.**
2. **Explain the importance of good record keeping and retention.**
3. **Demonstrate knowledge of proper ventilation measurement techniques.**
4. **Explain how branch resistance is calculated.**
5. **Utilize Kirchoff's First and Second Laws to balance quantity and pressure measurements.**
6. **Describe how to determine a fan operating point.**

Get Started

Go to the Course Index

About the Aeolus Project

## Course Summary:

| Date | Details |
|---|---|
| | Ventilation Surveys Quiz (https://canvas.instructure.com/courses/1095250/assignments/5490945) |

# 1.0 Quality Assurance for Mine Ventilation Surveys and Audits

A survey or audit of the ventilation system can refer to a range of activities and purposes depending on the context. This course will focus on the technical aspects of planning measuring and analyzing ventilation system parameters. Additional, conceptual information regarding context and purpose may be found in the course modules for “Health Effects, Measurement and Control of Diesel Emissions”, “Metal / Nonmetal Ventilation System Design” and “Ventilation Network Simulation and Modeling” attached to this series.

In practice, a ventilation survey involves the definition of the “current” or actual conditions in a mine; however, this only tells half of the story. In reality, a significant value of the ventilation survey lies in the input that it provides to the ventilation network model, allowing the accurate extrapolation of current system conditions into various future permutations. In this context, ventilation survey planning and execution must be given equal consideration (Rowland, 2010).

# 1.1 Ventilation Survey Planning

Survey planning in a significant portion of the actual "work" that occurs during a ventilation survey. Prior to ever going underground, the surveyor(s) should be familiarized with the maps, ventilation system and mine operating procedures. All pertinent safety precautions should be followed at all times. At a minimum, survey planners should consider the following:

- Surveyors should be familiar not just with all ventilation infrastructure, but with all evacuation routes and procedures, the locations of refuges, etc.
- Surveyors should be aware of mine blasting procedures, and of any specially-occuring events during the survey (e.g., equipment moves, shut downs, off-shift blasts, etc.).
- When possible, share your plan for the survey with the mine managers and shift supervisors for all areas you will visit ahead of time.
- Contact mine personnel when you enter an area of the mine for the first time.
- Measurements in production and hoisting shafts are possible, but will likely require additional planning and risk assessment.
- All accessible airways of the mine with appreciable airflows should be measured.
- All ventilation controls and fans (primary and booster) should be measured.
- Auxiliary systems may be measured depending on the scope of the study.
- Representative resistance factors (k-factors should be measured when possible.
- Plan the survey in order to minimize the impact the mine system and to capture an accurate "snapshot" of conditions.
- Data collected during the survey should be verified for internal coherence as often as practicable during the survey using Kirchoff's first and second laws.

Figure 1.1 shows a section of mine development to be surveyed. Figure 1.2 shows the plan for airflow measurements needed to define the airflow distribution. The planned pressure stations (start and end of tube-pulls for a gauge and pressure survey) are shown on Figure 1.3.

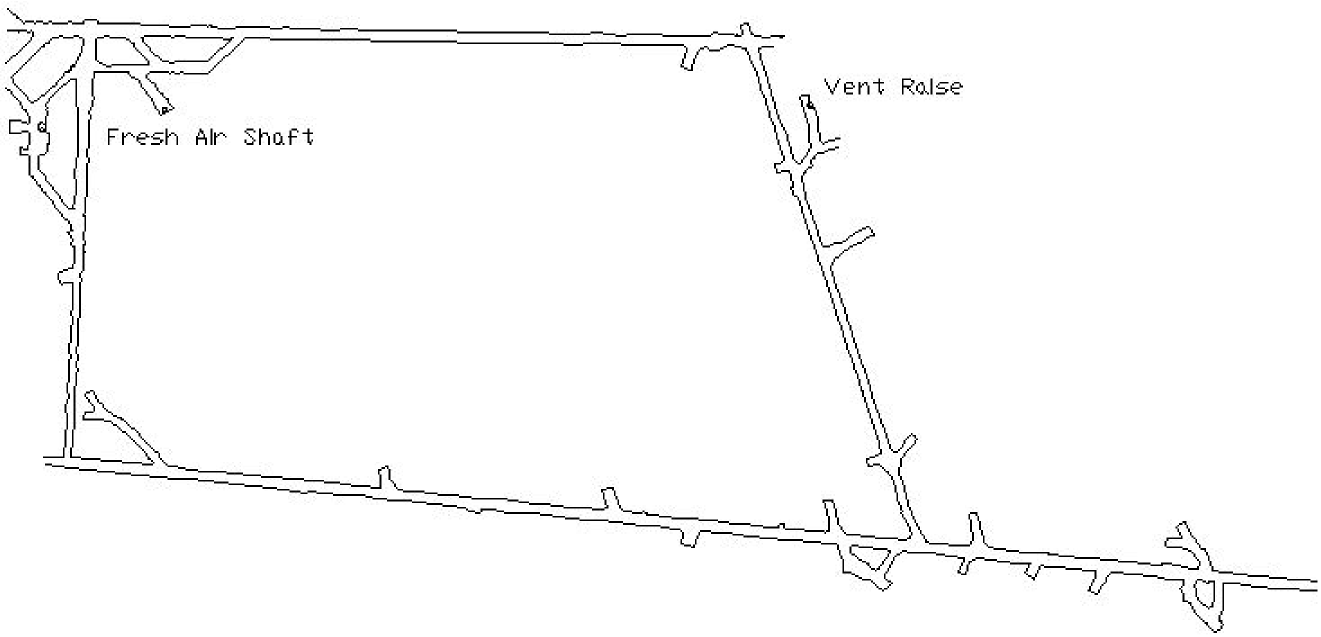



**Figure 1.1: Mine survey map showing mine development.**

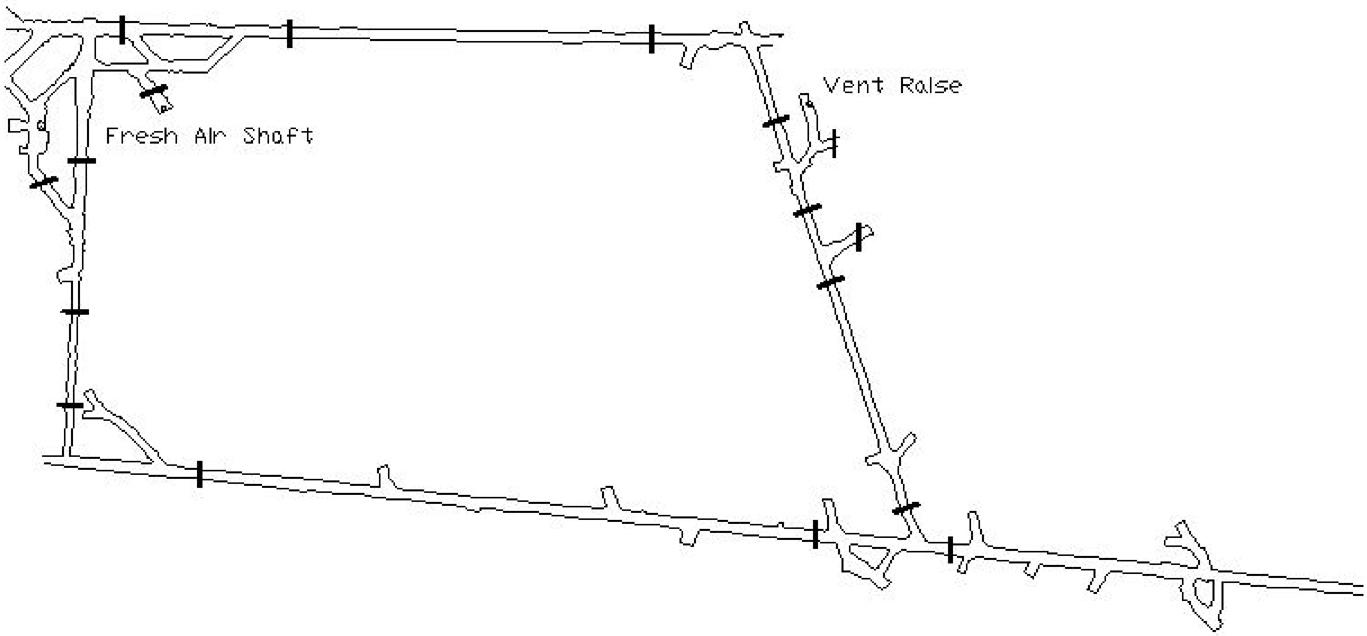



**Figure 1.2: Mine survey map with airflow measurement stations (planned).**

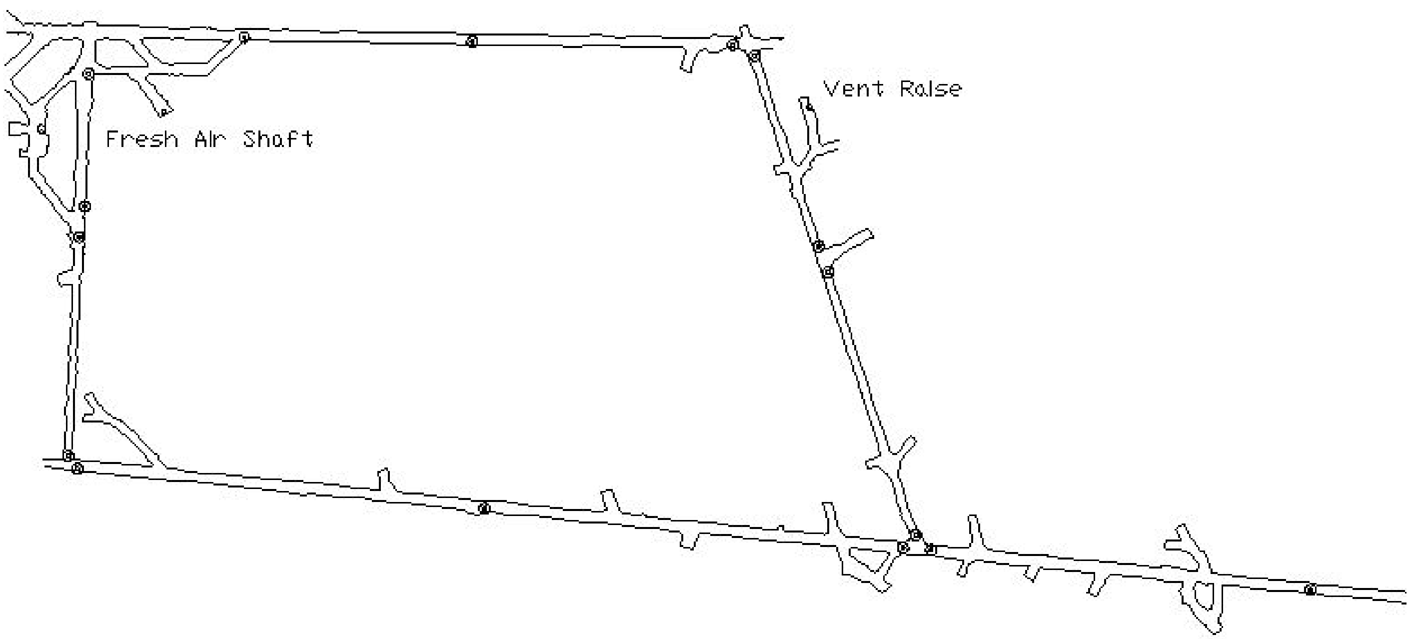
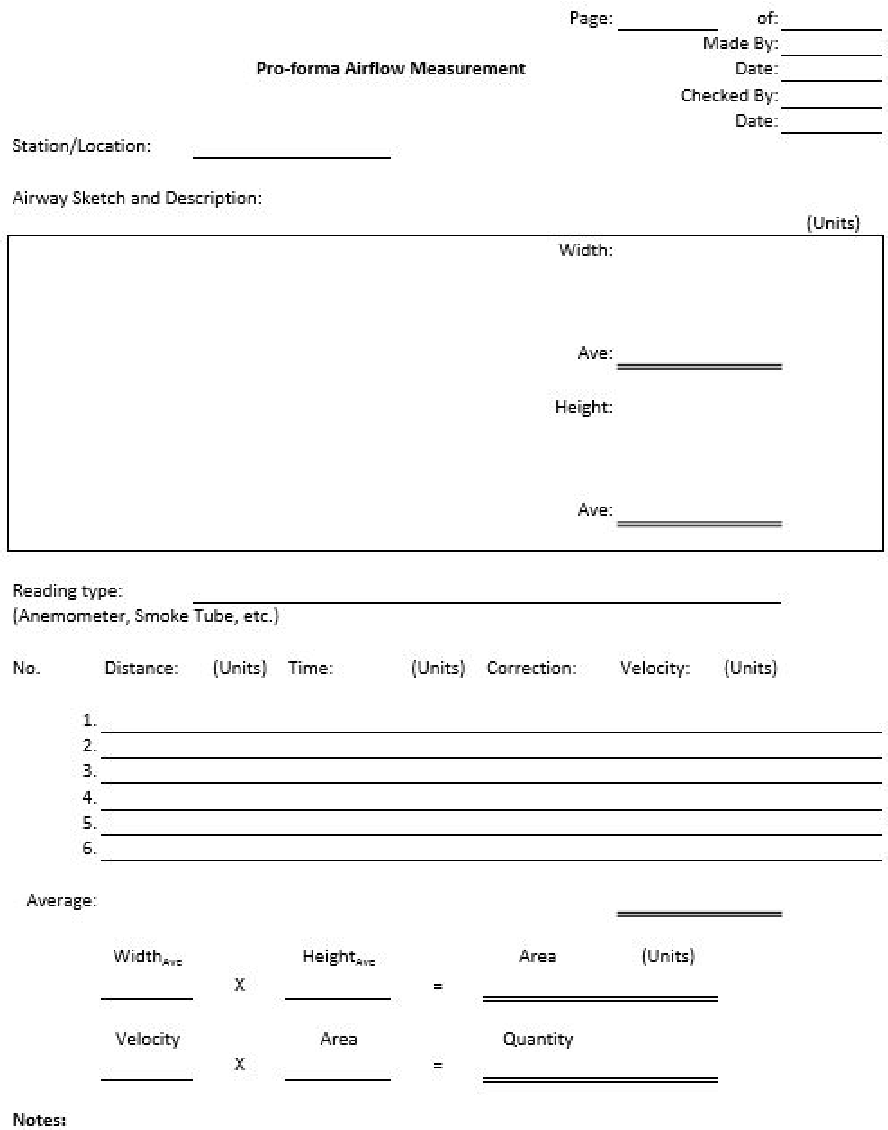



**Figure 1.3: Mine survey map with pressure measurement stations (planned)**.

# 1.2 Data Management and Record Keeping

As with most engineering projects, proper recording and storage of survey data is essential.  All survey data should be recorded in a consistent and repeatable manner and in as permanent a media as allowable. Consistency in recording your measurements and observations is important.  The use of pro-forma data collection sheets or books is recommended to assist in this matter.  Standard pro-forma for data collection are given in the following tables- following each table is a link to a pdf version of the pro-forma for download:

**Table 1.1 Airflow Quantity Survey pro forma.**



Page: ________ of: ________

**Pro-forma Airflow Measurement**

Made By: ________

Date: ________

Checked By: ________

Date: ________

Station/Location: ________________

Airway Sketch and Description:

(Units)

Width:

Ave: ________________

Height:

Ave: ________________

Reading type: ________________________________
(Anemometer, Smoke Tube, etc.)

| No. | Distance: | (Units) | Time: | (Units) | Correction: | Velocity: | (Units) |
|---|---|---|---|---|---|---|---|
| 1. | | | | | | | |
| 2. | | | | | | | |
| 3. | | | | | | | |
| 4. | | | | | | | |
| 5. | | | | | | | |
| 6. | | | | | | | |

Average: ________________

$\text{Width}_{\text{Ave}}$ X $\text{Height}_{\text{Ave}}$ = Area (Units)

Velocity X Area = Quantity

Notes:

**[Airflow Proforma.pdf](https://canvas.instructure.com/courses/1095250/files/46319037/download?wrap=1) (https://canvas.instructure.com/courses/1095250/files/46319037/download?wrap=1)**
**(https://canvas.instructure.com/courses/1095250/files/46319037/download?wrap=1) (https://canvas.instructure.com/courses/1095250**

/files/46319037/download?wrap=1)

**Table 1.2 Pressure Survey pro forma.**

Page: ________ of: ________
Made By: ________
**Pro-forma Air Density Measurement** Date: ________
Checked By: ________
Date: ________

Instruments Used: ________________

| | Station/Location: | Time: | $P_{barometric}$: (Units) | $T_{dry\text{-}bulb}$: (Units) | Humidity: (Units) | Velocity: (Units) | Notes: |
|---|---|---|---|---|---|---|---|
| 1. | | | | | | | |
| 2. | | | | | | | |
| 3. | | | | | | | |
| 4. | | | | | | | |
| 5. | | | | | | | |
| 6. | | | | | | | |
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| 13. | | | | | | | |
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| 22. | | | | | | | |
| 23. | | | | | | | |
| 24. | | | | | | | |
| 25. | | | | | | | |

**Notes:**

Pressure Proforma.pdf (https://canvas.instructure.com/courses/1095250/files/46319042/download?wrap=1) (https://canvas.instructure.com/courses/1095250/files/46319042/download?wrap=1) (https://canvas.instructure.com/courses/1095250/files/46319042/download?wrap=1)

**Table 1.3 Barometric/Psychometric pro forma.**

Page: ________ of: ________
Made By: ________
Date: ________
Checked By: ________
Date: ________

**Pro-forma Air Density Measurement**

Instruments Used: ____________________

| | Station/Location: | Time: | $P_{barometric}$: (Units) | $T_{dry\text{-}bulb}$: (Units) | Humidity: (Units) | Velocity: (Units) | Notes: |
|---|---|---|---|---|---|---|---|
| 1. | | | | | | | |
| 2. | | | | | | | |
| 3. | | | | | | | |
| 4. | | | | | | | |
| 5. | | | | | | | |
| 6. | | | | | | | |
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| 20. | | | | | | | |
| 21. | | | | | | | |
| 22. | | | | | | | |
| 23. | | | | | | | |
| 24. | | | | | | | |
| 25. | | | | | | | |

Notes:

**Density Proforma.pdf** (https://canvas.instructure.com/courses/1095250/files/46319045/download?wrap=1) (https://canvas.instructure.com/courses/1095250/files/46319045/download?wrap=1) (https://canvas.instructure.com/courses/1095250/files/46319045/download?wrap=1)

**Table 1.4 Fan Survey pro forma.**

Page: ________ of: ________
Made By: ________
**Pro-forma Fan Measurement** Date: ________
Checked By: ________
Date: ________

Fan/Location: ________________

Duct/Configuration Sketch and Description:

| | | |
|---|---|---|
| | $P_{barometric}$: | (Units) |
| | $T_{dry\text{-}bulb}$: | (Units) |
| | Humidity: | (Units) |

Gauge Used: ________________ Air Density (Units)

| No. | $P_T$ Ave.: | (Units) | Range: | $P_S$ Ave.: | (Units) | Range: | $P_V$ Ave.: | (Units) | Range |
|---|---|---|---|---|---|---|---|---|---|
| 1. | | | | | | | | | |
| 2. | | | | | | | | | |
| 3. | | | | | | | | | |
| 4. | | | | | | | | | |
| 5. | | | | | | | | | |
| 6. | | | | | | | | | |
| 7. | | | | | | | | | |
| 8. | | | | | | | | | |
| 9. | | | | | | | | | |
| 10. | | | | | | | | | |
| 11. | | | | | | | | | |
| 12. | | | | | | | | | |

**Notes:**

[Fan Proforma.pdf (https://canvas.instructure.com/courses/1095250/files/46319048/download?wrap=1)](https://canvas.instructure.com/courses/1095250/files/46319048/download?wrap=1) (https://canvas.instructure.com/courses/1095250/files/46319048/download?wrap=1) (https://canvas.instructure.com/courses/1095250/files/46319048/download?wrap=1)

Field data, including all measurements and observations should be transferred daily from field maps (underground) to clean maps (office). After the survey is completed, data should be retained for as long as practicable, but for a minimum of 10 years (in some format).

# 2.0 Airflow Quantity Surveys

Airflow quantity is a critical parameter governing and understanding underground ventilation systems. Airflow Quantity, as a concept, is easily described and understood by even those with no ventilation experience.  It is almost a given then, that the majority of ventilation-related legislation/regulation pertains to airflow quantity, or contaminant levels (that are directly proportional to airflow).

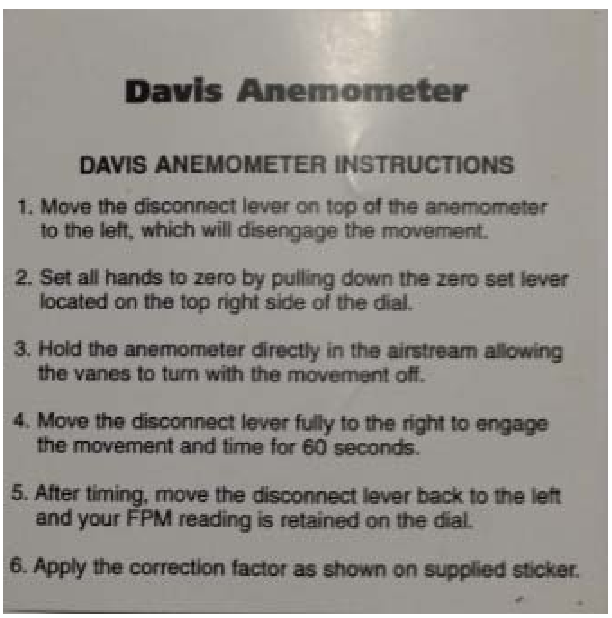
# 2.1 Vane Anemometers

Vane anemometers represent the most commonly used method for determining airflow quantity. They range in size, accuracy and quality, however; many acceptable types/manufacturers are commercially available for use in underground mines. This type of anemometer is both sufficiently accurate and robust enough to withstand the rigors of the underground mining environment.

The anemometer itself consists of a series of angled blades (similar to axial fan) that freely rotate around a hub. The anemometer is held perpendicular to the airstream, and records either the linear distance of air through the blades over a given time (analog) or mean velocity of the airflow (digital). In the case of analog anemometers, the reading must be taken over a known period of time (usually one minute, resulting in velocities with units of feet per minute or fpm). The reading is initiated and terminated through the use of a clutch actuated by a small lever attached to the anemometer hub. After the reading is obtained, the dial can be reset by the use another small lever. Each anemometer comes with a correction chart, which gives correction values based on the observed airflow velocity. When the exact measured velocity is not shown on the chart, the correction can be applied using simple, linear extrapolation.

Two people are required to obtain a velocity reading with an analog anemometer; the first operates the anemometer and performs the traverse, the second person times the measurement and enters data into the book or data sheet.

Figures 2.1 and 2.2 show the supplied instructions with an industry standard, 4-inch Ball-Bearing vane anemometer provided by Davis Instruments for operating the anemometer and determining airflow quantity, respectively. The anemometer is shown on Figure 2.3.


Davis Anemometer

DAVIS ANEMOMETER INSTRUCTIONS

1. Move the disconnect lever on top of the anemometer to the left, which will disengage the movement.
2. Set all hands to zero by pulling down the zero set lever located on the top right side of the dial.
3. Hold the anemometer directly in the airstream allowing the vanes to turn with the movement off.
4. Move the disconnect lever fully to the right to engage the movement and time for 60 seconds.
5. After timing, move the disconnect lever back to the left and your FPM reading is retained on the dial.
6. Apply the correction factor as shown on supplied sticker.

**Figure 2.1: Davis Instruments supplied instructions for vane anemometer use.**

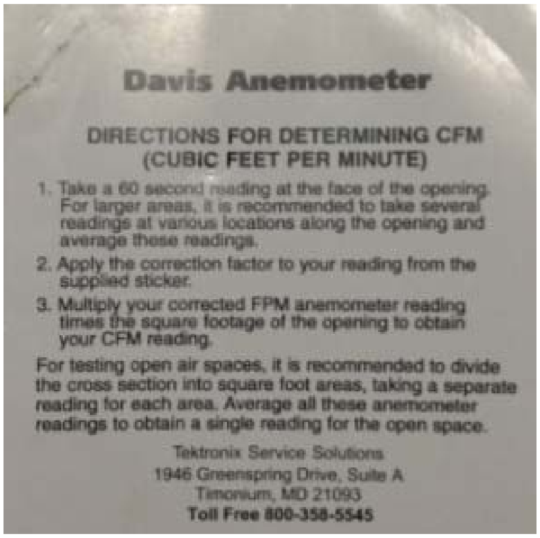



**Figure 2.2: Davis Instruments supplied instructions for volume flow rate determination.**

**Figure 2.3: 4-inch Ball-Bearing vane anemometer (Davis Instruments).**

When using a digital electronic vane anemometer, the process is much the same, but with buttons depressed in place of mechanical levers for stopping, starting and resetting the device. An electronic vane anemometer is shown for reference on Figure 2.4. One of the advantages of a digital anemometer that displays a mean velocity as an output is that a second person (the timer) is no longer needed. As long as the anemometer reading is performed at an appropriate speed, then the exact amount of time required to traverse the drift is not critical, and the velocity is always recorded in units of units distance/unit time regardless of the length of the measurement.

**Figure 2.4: A 4-inch digital electronic anemometer with backlit screen (Testo).**

For ventilation surveys, the anemometer is attached to an extension rod (>5 ft) that extends the cross-section of the measurement out away from the body of the observer (and outside any influence of the person on the airstream velocity).  It is important to maintain the anemometer in a plane perpendicular to the airstream and to consistently measure in that cross-section of the drift. When the airflow velocity is obtained, it is then multiplied by the cross-sectional area to obtain a volume flow-rate (quantity of air).

$$Q = v \times A$$

Where:

Q = Airflow quantity (cubic feet per minute or cfm)

v = velocity (fpm)

A = cross-sectional Area (square feet)

In practice, the anemometer traverses in a serpentine fashion from a point midway up one side of the tunnel to the other (in a consistent cross-section of the drift).  Generally, at least two velocity measurements are taken in opposite directions at each survey location provided that they correlate to within 5% of each other.  In some cases, it may be necessary to repeat the velocity measurements until sufficient correlation is achieved.

Figure 2.5 shows the path taken by the anemometer during a typical velocity reading.

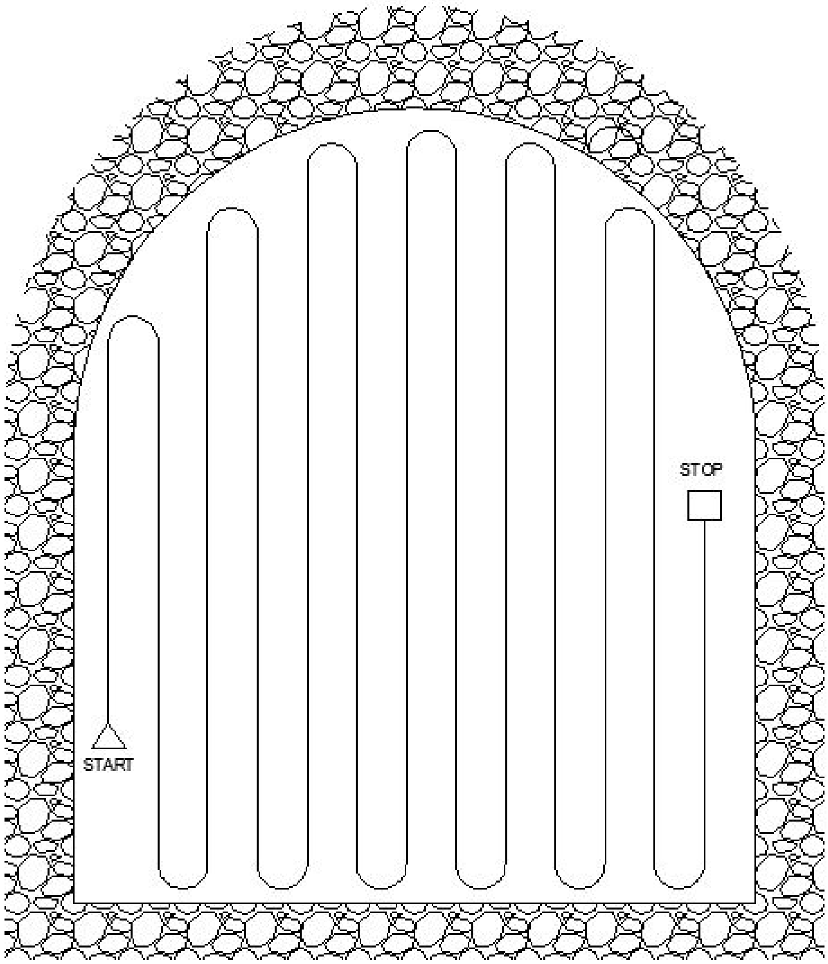



**Figure 2.5: Recommended anemometer traverse path.**

Practically, best results are achieved when velocity readings (measurement stations) are located at least two hydraulic diameters upstream or eight hydraulic diameters downstream of and significant, bends, intersections, obstructions or other changes to the velocity profile of the drift (even though this is not always possible).

Data recorded during the reading includes the location/number of the measurement, the drift profile (with width and height measurements), the direction of airflow and the average velocity of the air stream.  In the absence of surveyed cross-sectional areas, two widths (about knee and shoulder height) and three heights (at the top of each rib and at the drift centerline) are measured with a laser-distance meter (preferred) or a measuring tape.  A sketch of the drift profile, with any obstructions, services or other notes regarding the conditions observed should also be included.

Figure 2.6 shows the survey map with measured airflow quantities recorded.  Note that actual survey locations may differ slightly from planned measurement stations.  This can be due to a variety of factors, including errors in map making, equipment locations, stored materials, etc.

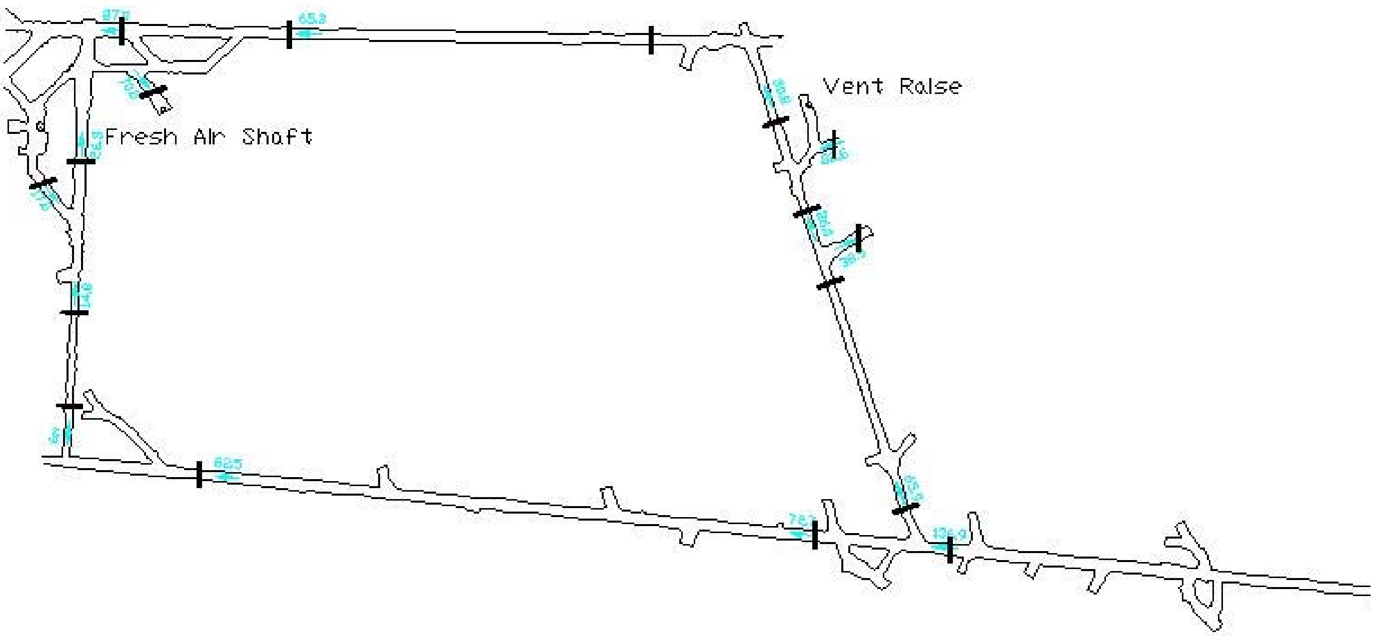



**Figure 2.6: Airflow quantity survey data recorded on map.**

# 2.2 Hot-wire Anemometers

Hot-wire anemometers work on the principal of a wheatstone bridge, where one arm of the circuit is comprised of the recording element.  As heat is removed from the element by the airstream moving past it, this energy loss is measured by the device.

Hot-wire anemometers, while not particularly suited for general use in measuring airflows for a full ventilation survey, are particularly well suited to taking spot measurements, duct measurements (for defining auxiliary ventilation systems) and for establishing velocity profiles in ducts and drifts.

To obtain a direct velocity reading in a ventilation duct, the probe is simply inserted through a small hole until it is located approximately in the center of the airstream, and the velocity can be read directly from the gauge.

If a hot-wire anemometer is to be used to obtain an airflow quantity in a larger drift or tunnel, then a grid traverse should be performed.  In order to account for changes in the velocity profile of the drift or entry, it is divided into a grid, with an average velocity reading taken at each junction or intersection.

Figure 2.7 shows a typical grid for obtaining an airflow by taking spot-readings with a hot-wire anemometer or Pitot tube.

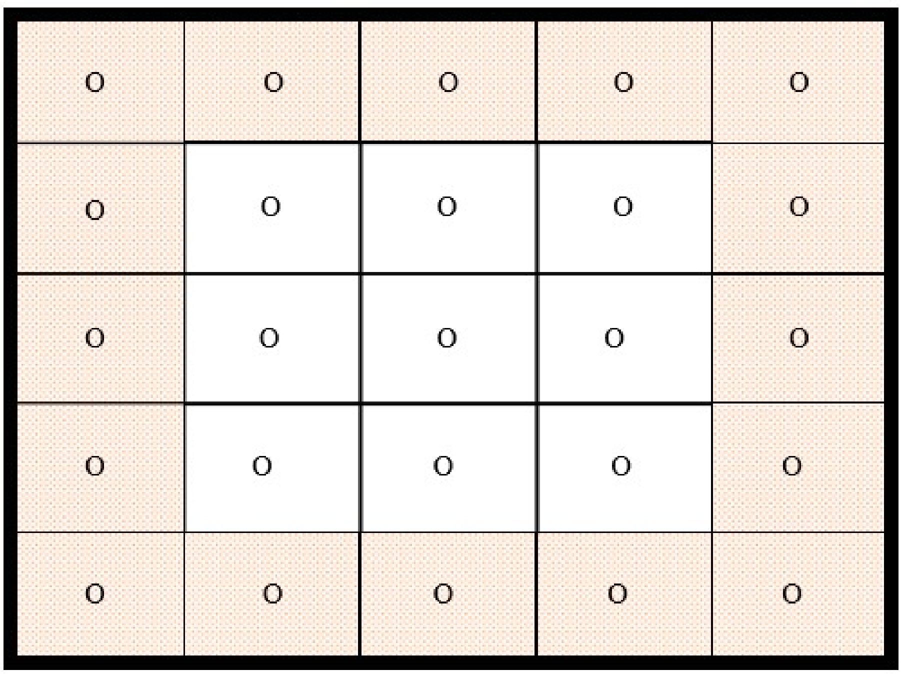


**Figure 2.7: Recommended grid for taking spot readings of airflow velocity in a large-diameter entry (McPherson).**

# 2.3 Pitot Tubes (Velocity Pressure)

Although directly used for conducting pressure measurements (when connected to a manometer), Pitot tubes can also be used to accurate determine airflow velocities in support of calculating airflow quantities provided that the density of the airstream is known (or can be calculated from concurrent measurements of barometric pressure, temperature (dry-bulb) and either wet-bulb temperature or relative humidity.

When held such that the air stream flows into the open end of the Pitot tube, the velocity pressure can be read on a pressure gauge (manometer) by obtaining the difference between the total pressure (center tube) and the static pressure (outer tube).

The velocity of the airstream is determined by the following equation:

$$v = 1,096\sqrt{\frac{p_v}{\rho}}$$

Where:

$v$ = airstream velocity (fpm)

$p_v$ = velocity pressure (inches water gauge)

$\rho$ = the density of the airstream ($lb/ft^3$)

Figure 2.7 shows a cross-section of a standard Pitot tube (McPherson). Figure 2.8 shows an actual pitot tube attached to a digital micro-manometer (pressure gauge).

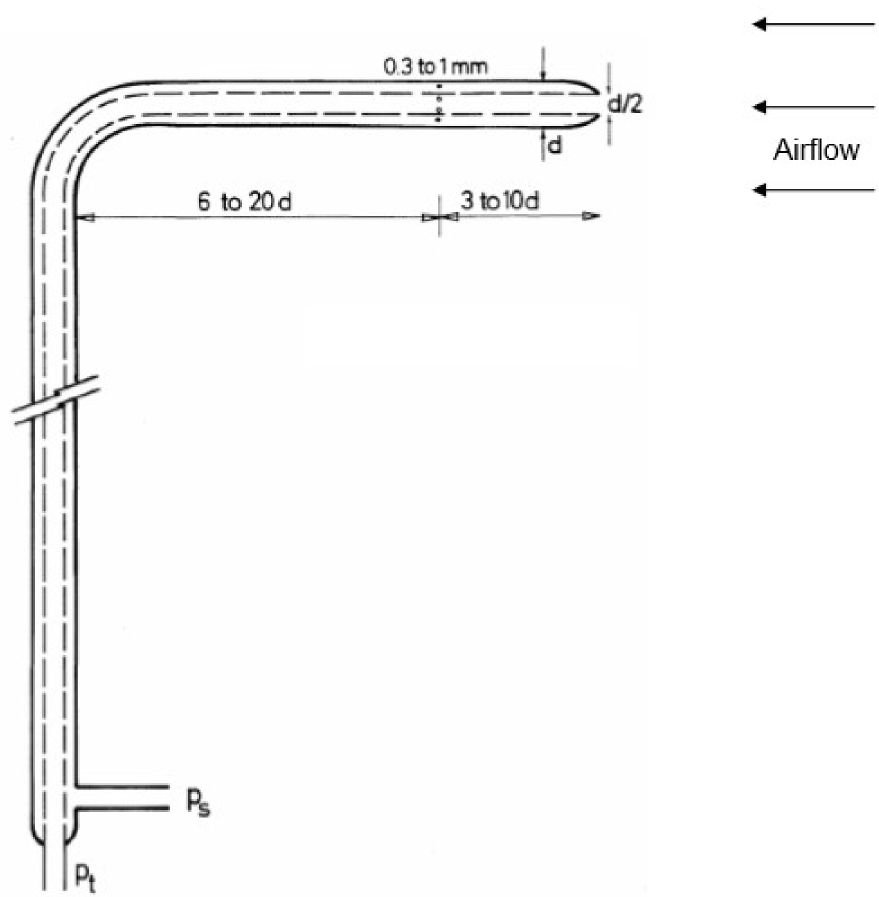


Figure 2.7: Cross-section of a standard Pitot tube (McPherson).


**Figure 2.8: Pitot tube attached to a digital pressure gauge.**

Pitot tubes are useful in any scenario where a spot reading of velocity is required, similar to that of a hot-wire anemometer.

# 2.4 Smoke Tubes (Low Velocity Measurements)

In air velocities less than 50 fpm (and in some other situations), smoke tubes may be used to measure the speed and or direction of airflow.  In such conditions, neither a vane anemometer nor a Pitot tube will provide an accurate velocity reading.  In order to obtain a velocity reading with smoke tubes, a precise distance (usually 10 ft) is delineated with the measuring device (laser or tape) in the approximate centerline of the drift.  A puff (or puffs) of smoke are released at one end of the measurement at the same time as the timer is started.  When the smoke reaches the end of the measured distance the timer is stopped.  The velocity of the smoke can then be determined:

$v_{smoke} = \frac{10\ ft}{x\ \text{sec}} \times 60\frac{\text{sec}}{\text{min}}$ (fpm)

Because this represents only the velocity of the air in the center of the drift, it should be multiplied by a factor between 0.6 and 1 in order to represent the total velocity profile over the entire drift.  A typical value of 0.85 is most often used to correlate centerline measurements of velocity and an average for a full cross-section.

# 3.0 Pressure Surveys

Although much less well understood (even by some ventilation practitioners), the differential pressure losses in an underground circuit are critically important both to understanding the airflow distribution in an underground circuit and to determining the resistance to flow necessary to build an accurate ventilation network model. Generally, frictional pressure losses in mines are measured using one of two methods in practical mine ventilation surveys.

The "Gauge and Tube" method, involves the direct measurement of pressure differential through the use of a small hose, connected in-line with a digital micromanometer and two pitot tubes, allows for the rapid (and accurate) determination of pressure loss between the two endpoints of the tube, and has the further advantage of allowing immediate evaluation for internal coherence of data through Kirchoff's Second Law.

The other common method for measuring differential pressure losses in underground mines utilizes a combination of two barometers to measure the relative barometric pressures at two points along with a host of other data. This "Barometer Method" also involves the collection of other data required for the calculation of differential pressure, and involves more complex mathematical calculations that generally cannot be performed in the field in order to provide immediate correlation of field data. Further complicating the reduction of data in the Barometer Method are the various different equations or protocols that exist, which often give vastly different results, making correlation of field data, or comparisons between projects with different protocols difficult or impossible.

# 3.1 Gauge and Tube or Trailing Hose Method

The “Gauge and Tube” method of measuring differential pressures involves the direct and immediate measurement of total or static pressure losses between two points (corresponding to the start and end points of a branch in the mine ventilation network model.

It is critically important that the two points be located in the same quantity (volume flow) of air- thus all direct pressure measurements will begin after one intersection and end prior to the next branch junction. It sometimes occurs that there will be parallel paths in between the two endpoints- and this is acceptable given that the pressure drops across the two parallel paths will be equivalent (Kirchoff’s Second Law).

Static pressure differentials can be measured directly across bulkheads and regulators (assumes that the airflow velocity is very low- effectively zero) across relatively short distances.

In the case of longer measurements (up to 1,000 ft) in open entries, the differential pressure is simply defined as the difference (or delta) in total pressure between the two measurement points- in most cases, no additional calculations are required (McPherson, 2009).

Practically, this is accomplished as follows:

1. The lead surveyor advances from the first observation station (Point 1) holding the free-end of the tube, a pitot-tube and the pressure gauge to the second station (Point 2) or the end of the tube, whichever is reached first.
2. The hose and the Pitot tube are attached to the micromanometer, and the Pitot tubes on either end are held in the center of the airstream such that air flows into the tip of the Pitot tube.
3. A period of time (approximately five minutes) is required in order for the pressure reading to stabilize. The pressure reading is obtained by reading the differential pressure between the trailing hose and the local Pitot tube (attached to the gauge via short piece of flexible, vinyl hose).
4. Once the reading is recorded on the field map (time, start point, end point, direction of flow, average pressure differential, minimum and maximum), the next station is marked by paint, or survey flagging tape on the rib, roof or mine services (air, water, etc.). Across long sections of drift, this will be at the exact location where the previous measurement was taken; however, where the measurement ends at an intersection, the next measurement station will lie across the junction.
5. The rear-tubeman is then alerted via radio, cap-lamp or via tugging on the tube according to a pre-agreed signal and the surveyor advances with the rear of the tube following behind. When the next station is reached, the rear-tubeman stops and tugs the tube securely to alert the lead surveyor.
6. This process is repeated until all planned measurements have been completed.

Figure 3.1 shows the actual pressure measurements recorded during a pressure survey (gauge and tube).

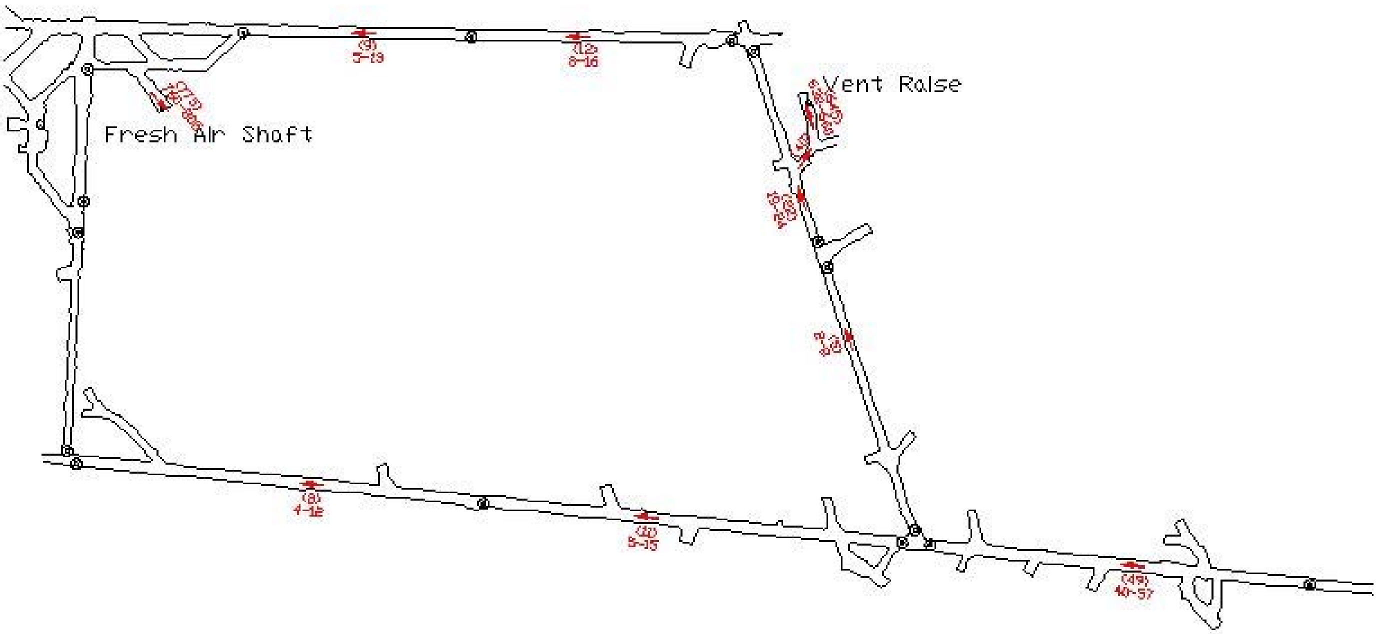



**Figure 3.1: Schematic of actual pressure measurements performed with a gauge and tube.**

Figure 3.2 shows a pressure measurement being performed with a gauge and tube.

**Figure 3.2: Pressure measurement being performed with a gauge and tube.**

Figure 3.3 gives an example of a commercially available digital micro-manometer.


**Figure 3.3: Digital micro-manometer (Fluke).**

The following equipment is required for a pressure survey:

- ¼ inch nylon tube – 500 to 1,000 ft on hose reel (nylon has been demonstrated to be sufficiently flexible, durable and cost-effective for performing these measurements in mining environments).
- Digital micromanometer
- 2 Pitot tubes (3 – 4 ft)
- ¼ inch vinyl (flexible) tube – 6 to 10 ft for measuring across doors, regulators, etc.
- Welding tip cleaners or paperclip (helpful for cleaning Pitot tubes)
- Electrical tape for sealing the hose end and Pitot tubes when appropriate.
- Spray paint and/or surveyor's flagging tape for marking survey stations
- Tube couplings for performing field repairs
- Maps of the area(s) to be measured.
- Pen(s)
- Knife or tube cutter

**Notes:**

- **Modern, digital micromanometers provide accurate, easy to read measurements to 5-thousandths of an inch of water gauge (1 Pascal) or better over the range of differential pressures commonly encountered in mines.**
- **Air flows from areas of high pressure to low pressure, indicating which tube (front or rear) should be connected to which port on the gauge.**
- **Care must be taken to prevent the intrusion of water, mud and excessive dust into the hose or the pitot tubes.**
- **Measurements of barometric pressure, dry-bulb temperature and wet-bulb temperature or relative humidity should be taken periodically or at any time when the air density can reasonably be expected to change.**

# 3.2 Barometer Method(s)

Utilizing a pair of accurate barometers to determine the pressure differential between two points has been a commonly utilized technique in underground mines for decades of practice. This method requires two extremely accurate instruments, and the coordination of at least two measurement teams. The calculations required to determine the differential pressure (from measurements of ; however, it does have the advantage that direct access is not required for measurement, making it particularly well suited for areas that are inaccessible to either physical location or unsafe conditions.

Several methods for the reduction of barometric survey data exist, including the following:

$$F_{12} = \frac{u_1^2 - u_2^2}{2} + (Z_1 - Z_2)g - R(T_2 - T_1)\frac{\ln(P_2/P_1)}{\ln(T_2/T_1)}$$

**Steady-Flow Energy Equation Method (McPherson)**

The first method used to determine the pressure loss in the FAR involves the direct application of the steady-flow energy equation as outlined by McPherson in “Subsurface Ventilation and Environmental Engineering”.

Where:

F = Work done against friction (J/kg)

P = Barometric pressure (kPa)

T = Absolute temperature (Kelvin)

Z = Elevation of barometer location (m)

u = Air velocity at the barometer location (m/s)

R = Mean gas constant (J/kg K)

g = Gravitational acceleration (9.81 m/s$^2$)

The frictional work (F) is then converted to frictional pressure loss via the following equation:

$$p_{12} = \rho_a F_{12}$$

Where:

$p_{12}$ = Frictional pressure drop (Pa)

$\rho_a$ = Average density of air between two stations (kg/m$^3$)

If the two measurement locations are not read simultaneously, it is necessary to apply a correction to one of the barometric pressure readings in order to account for changes in the atmospheric condition in the time between the readings. This is done according to the following formula:

$$P_1' = \Delta P_c \frac{P_1}{P_c}$$

Where:

$P_1'$ = Updated value for barometric pressure at station 1

$P_1$ = Raw data for barometric pressure at station 1

$DP_c$ = Change in surface atmospheric pressure

$P_c$ = Surface atmospheric pressure taken at the same time as station 1 reading

**Mine Ventilation Society of South Africa (MVSSA) Method**

The following equation for determining the pressure loss between two points is advocated by the esteemed MVSSA:

$$p_{12} = -(P_2 - P_1) - g\int wdZ$$

Where:

$P_1$ = Barometric pressure at station 1

$P_2$ = Barometric pressure at station 2

The term $g\int wdZ$ in this case is the difference in theoretical pressure (head). There is some difficulty in evaluating this component to account for the change in air density between measurement stations resulting from the difference in elevation. To simplify the calculations, it is assumed that the density between the two stations varies linearly with respect to elevation. The integral is evaluated according to the following equation:

$$\int wdZ = \frac{1}{2}(\rho_1 + \rho_2)(Z_1 - Z_2)$$

Where:

$\rho$ = air density

$Z$ = elevation

This method is considered acceptable for situations where the elevation change between stations is less than 300 m (McPherson, 2009). Errors associated with this method increase when the elevation difference is large and when condensing environments exist between stations.

**Exact Density Method (C.J. Hall)**

The following method for reducing barometer survey data comes from C. J. Hall, as published in “Mine Ventilation Engineering”. The exact density solution as explained by Hall assumes a linear change in density with depth as in the MVSSA method; however, Hall extends the acceptable change in elevation between stations to 700 m for his solution. In this case, a “frictionless pressure” is calculated from known parameters, and then used to determine the pressure loss between stations according to the following equation:

$$p_{12} = P_{2calc} - P_2$$

Where:

P2calc = "Frictionless pressure"

P2 = Barometric Pressure at Station 2

And:

$$P_{2calc} = P_2\left(\frac{2P_1 + D_1 g\,\rho_1 10^{-3}}{2P_2 - D_2 g\,\rho_2 10^{-3}}\right)$$

Where:

$\rho$ = Air density

D = Depth below datum

**CASE STUDY – Comparison of methods for determining the pressure loss in a Fresh Air Raise**
(https://canvas.instructure.com/courses/1095250/files/52627651/download?wrap=1) (https://canvas.instructure.com/courses/1095250/files/52627651/download?wrap=1) (https://canvas.instructure.com/courses/1095250/files/52627651/download?wrap=1)

# 4.0 Determining Air Densities

Whichever method is used to determine the differential pressure, the following information is needed for the calculation of air density and Natural Ventilation Pressure (NVP):

- Barometric Pressure ($P_b$)
- Dry Bulb Temperature ($T_{db}$)
- Wet Bulb Temperature ($T_{wb}$) or Relative Humidity (RH)

### Barometric Pressure

Barometric pressure is one of the key parameters affecting air density in underground environments.  Although it is not necessary to obtain the same level(s) of accuracy in measurements of barometric pressure for the purposes of calculating NVP as it is to calculate resistance, the same barometer may be used in both cases for the purpose of minimizing the equipment required to be carried underground.

There exist several manufacturers and types of barometers available that are sufficiently accurate for use in obtaining atmospheric pressure measurements for the purpose of determining NVP, including Kestrel, GE/Druck and Vaisala, etc.  Prices generally range from a few hundred to a few thousand dollars, depending on the accuracy and features included in the individual units.  All measurements should be recorded to at least the nearest hundredth of a kilopascal (e.g. 96.75 kPa).

### Temperature (Dry Bulb)

Although traditional measurements of temperature were made by sling-psychrometer, the existence of highly accurate and reliable digital thermometer gauges/meters means that the old days of whirling a spinning thermometer above one's head, then trying to read it by the light of a cap lamp before it changed are over.  The new digital gauges are easier to read, more robust, and more user friendly than old sling psychrometers and in practice, provide better data.  Measurements of temperature should be recorded to a tenth of a degree Celcius (e.g. 21.6 °C).

### Temperature (Wet Bulb) or Relative Humidity

The choice of measuring wet bulb temperature or relative humidity in support of air density calculations is often debated by ventilation engineers and practitioners.  Although it is generally agreed that conducting such calculations according to first principals is to be preferred, practical experience has shown that it is just as possible to make accurate determinations of NVP based on air densities calculated from measurements of relative humidity as from those based on wet bulb temperature.  For these reasons, it is recommended to measure the relative humidity of the air to the nearest tenth of a percent (e.g. 55.5%) as part of any psychrometric survey.

Once the barometric pressure, temperature and relative humidity of the air has been measured, it is possible to calculate the air density from the psychrometric equations (McPherson, 2009).

If the wet bulb temperature is known, the density can be determined as follows:

$$\rho_{actual} = \frac{(P - 0.378e)}{287.04(t_d + 273.15)}$$

where:

P = Barometric Pressure

e = saturation vapor pressure

$t_d$ = dry-bulb temperature

If relative humidity has been measured, the following relationship is used to determine e:

$$e = 610.6exp\left[\frac{17.27t_w}{237.3+t_w}\right]$$

where:

$t_w$ = wet-bulb temperature

If relative humidity has been measured, substitute the following for e

$e_{sw} = e_{sd} \times RH$:

Which yields the following:

$$\rho_{actual} = \frac{\left(P - 0.378\left(RH\left(610.6exp\left[\frac{17.27t_d}{237.3+t_d}\right]\right)\right)\right)}{287.04(t_d + 273.15)}$$

**Notes:**

- **Measurements of barometric pressure and temperature should be made in SI Units in order to simplify the calculations and minimize the introduction of rounding errors.**
- **The time of day and the location of the measurements should also be recorded at each location.**
- **In order to calculate the air density in Excel, use the following equation (pasted directly into the formula line):**

  **=(A3*1000-(0.378*(C3/100*(610.6*EXP((17.27*B3)/(237.3+B3))))))/((287.04*(B3+273.15)))**

  **where:**

  **A3 = Barometric Pressure in kilopascals**

  **B3 = Dry-bulb temperature in Celcius**

  **C3 = Relative Humidity in percent**

# 4.1 Natural Ventilation Pressure (NVP)

Although it is not necessary to measure differential pressure losses or airflow quantity in order to establish the NVP acting on a mine ventilation system, psychrometric readings are usually taken during a mine ventilation survey. One important consideration in the planning of a survey is that if the gauge and tube method is utilized in the measurement of differential pressures in the mining circuit, then additional equipment and measurements will be required for the quantification of airflow density required to determine NVP. If a barometer survey is conducted at a mine, then it is likely that all measurements required for the determination of NVP will have been completed during the required survey measurements. It should be noted that this is not, in itself, an argument for, or endorsement of the barometer survey technique for determining differential pressure losses in underground mines.

If we consider the classic example of a U-tube manometer, or to put it even more simply, two columns of fluid connected across the bottom; it serves as an able, if simplified analogue to an underground mine. In this case, it is relatively easy to visualize the displacement or movement of the fluids caused if there is an imbalance in the density between the two. In the case of most mines, this imbalance in density (since the fluid is the same throughout) is caused by the addition of heat from strata, mobile equipment, blasting, etc. In this case, the airflow is induced (or in the case of mines with mechanical ventilation systems, aided or retarded) by the difference in the mean density of air in the shafts and the depth of the shafts. The pressure differential that results in the movement of the air (NVP) as outlined by McPherson in “Subsurface Ventilation and Environmental Engineering” is equal to:

$$\mathrm{NVP} = \rho_1 (\mathrm{Z}_1 - \mathrm{Z}_2)\mathrm{g} - \rho_1 (\mathrm{Z}_1 - \mathrm{Z}_2)\mathrm{g}$$

Where:

ρ = Work done against friction (J/kg)

Z = Elevation of shaft top ($Z_1$) and Bottom ($Z_2$) (m)

g = Gravitational acceleration (9.81 m/s2)

In order to calculate more complex systems such as those found in most underground mines, a thermodynamic analysis of natural ventilating energy (NVE) may be performed by summing the integrals of volume with respect to pressure through each branch of the circuit (some individual branches within the mine may be treated as a single branch in the calculation):

$$NVE = \int VdP$$

which is differentiated from the NVP by the mean density of the air:

$$NVP = \rho_{mean} \int VdP$$

Where:

$g\int wdZ$ = area enclosed in a PV diagram between intake and returns-

this area is the Natural Ventilation Energy (NVE) (ft. lbf/lb)

$\rho_{mean}$ = mean density of air in intake and return airways (lb/ft$^3$)

Some ventilation network simulation programs include calculations for determining NVP internally. In ventilation models based on the assumption of incompressible flow, the NVP is usually represented by a fixed pressure fan, located in the exhaust surface connection(s). Because the pressure is not, in actuality applied at a single point of the mine atmosphere, but is in fact applied over the entire circuit, some further reduction of the data may be necessary to obtain an accurate distribution of NVP within the model. Generally, this is done by solving for the NVP of multiple overlapping loops within the system and solving for the pressure differential in each by the same number of equations as variables or through the matrix method of equation solving.

Despite the fact that NVP often has a negligible effect on the ventilation system of many mines, a working knowledge of how to measure for and calculate NVP is essential to the understanding of ventilation pressure surveys of subsurface environments. Without consideration of the NVP in the measurement and reduction of pressure losses throughout the ventilation system, an accurate model of the mine ventilation system cannot be reliably compiled and maintained. In cases where the NVP meets or exceeds 5% of the total fan pressure, it should always be accounted for in the ventilation model, regardless of whether or not the network simulation is performed with a program assuming compressible or incompressible airflow.

**Notes:**

- **A positive value for NVP represents a ventilation pressure assisting the fan(s).**
- **A negative NVP represents a ventilation pressure opposing the fan(s).**
- **The mean air density used in the calculations is simply the arithmetic mean of the densities at the top and bottom of the shaft(s) from the measured survey data.**
- **If a ventilation network simulation program is being used to determine the NVP, ensure that all relevant input parameters are correct!**
- **NVP represents a seasonal condition and will change as the atmospheric conditions change. In some cases, the temperature of the ambient air entering the mine will fluctuate from lower than the rock strata temperature to higher-reversing the effect of NVP from helping to hindering the mine ventilation fans or vice-versa.**

**CASE STUDY – NVP calculations for a North American Metal Mine – winter condition.** (https://canvas.instructure.com/courses/1095250/files/52627655/download?wrap=1) (https://canvas.instructure.com/courses/1095250/files/52627655/download?wrap=1) (https://canvas.instructure.com/courses/1095250/files/52627655/download?wrap=1)

# 5.0 Fan Surveys

Theoretically, the measurement of airflow quantity, pressure differential and air density for fans is no different than that in drifts. The basic processes for measurement, and the equations required for calculations have already been addressed in Sections 2.0, 3.0 and 4.0, respectively. However, in practice fan surveys have several unique considerations, and the procedure for performing the basic measurements required for the quantification of fan operating points are presented here in further detail.

Additional information about fans and fan measurements can be found in the “Fans” course.