NARAYANI INSTITUTE OF ENGINEERING & TECHNOLOGY
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4th Semester, Mining Engineering

Theory – 3

Sub: - Mine Ventilation

Chapter – Leakage of air in Mines

Leakage Through Mine Stoppings

Introduction

It is wasteful enough to operate against a high mine resistance, but far more wasteful is to handle a large volume of air of which only a small percentage is actually reaching the working areas. In operating mines today, it is not uncommon to lose between 60 to 80 percent of the air between the fan and the last open cross-cut due to air leaking directly into return airways.

Fugitive air losses as a result of poorly maintained stoppings and overcasts will cause shortage of fresh air at working sections where workers need more fresh air and where the major job of diluting and carrying away gases and dusts is conducted Furthermore, in order to compensate for these losses, additional air has to be handled at the fan. This will not only cause dust problems in airways due to higher velocities within the ventilation system, but will also increase power costs.

Unfortunately, all the control devices are subject to natural deterioration over time, mainly caused by strata convergence and damages (vehicles running into stoppings and overcasts blasting underground), both of which will result in increased air leakage. As a matter of fact, the majority of air leakage in underground coal mines are happening at or near the bottom of the slope where the pressure differential is the highest, the control devices are the oldest.

Air leakage problems have been around for a long time. For example, survey results from sixteen mines by Montgomery in 1936 showed that on the average only 19 percent of the air handled by the fan reached the last open cross-cuts, the rest of fresh air has leaked through stoppings, doors, and overcasts directly into return airways. It was also reported in 1955 that the first half of the ventilation circuit contributing about 75 percent of the total loss.
Another ventilation survey of 22 coal mines in the Central Basin in U.S. conducted in 1952 concluded that the percentage of air losses due to leaky stoppings, trap doors, and overcasts varies between 8.9 and 70.7 percent. In the worst case, only 29.3 percent of the fan quantity reached the active workings.

<u>Brattice Window Method</u>

The purpose of this study is to familiarize the students with the concept of leakage through a stopping and to determine the amount of such leakage through the use of the Brattice Window Method as described as follows.

The procedure is a simple one. A second stopping, called the temporary test stopping (TTS), is erected in the same entry as the leaking permanent stopping (see figure below).
The TTS is made of an impervious fabric, such as plastic mine brattice, and is fastened to the roof, floor, and sides of the entry with spads or similar fasteners. The TTS also will leak, as air will pass through gaps around the edges.

A rectangular opening, window 1, is cut into the TTS.
The cross-sectional area of this window and the velocity of air passing through it are measured and the volume flow calculated from
$Q_1 = V_1 A_1$,

where A = cross-sectional area of window 1, ft^2,
V = air velocity through window 1, fpm,
and Q = air volume through window 1, cfm.

Next, a second rectangular opening, window 2, is cut into the TTS.
Its area A_2 and the velocity of air through it V_2' are measured and used to calculate the air volume Q_2' through it.

$Q_2' = V_2' A_2$,

the decreased air velocity V_1' through window one is also measured and a new lower air volume Q_1' is calculated from

$Q_1' = V_1' A_1$

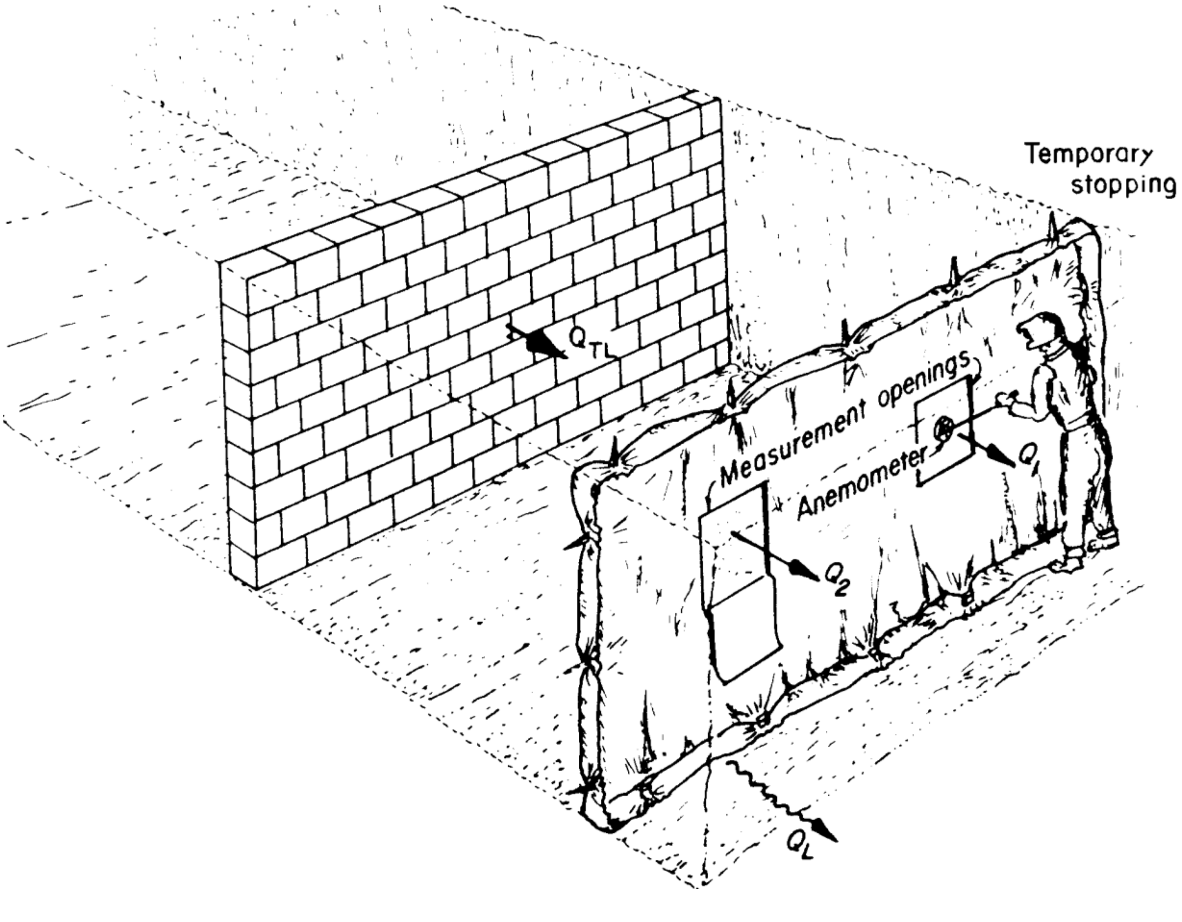

Fig: Leakage through Mine Stoppings

These values are used in the brat/ice-window-method equation to calculate the total volume of air QTL in cfm passing through the permanent stopping as follows:

$$Q_{TL} = 0.82\left[Q_1' + Q_2' + \frac{Q_1' + Q_2' - Q_1}{V_1/V_1' - 1}\right]$$

The 0.82 window correction factor is necessary because of the vena contract created by the airflow through the windows.

The last term of the equation, $\frac{Q_1' + Q_2' - Q_1}{V_1/V_1' - 1}$

is the "leakage" term which gives the total volume of air leaking around the TTS.

Procedure

1. Meet at the mine classroom to discuss the lab.
2. Close all temporary stoppings.
3. Install the Test Temporary Stopping (TTS) and prepare it for the Brattice Window Method experiment.
4. Turn the mine fan on low speed.
5. Take the necessary readings with the vane anemometer.
6. Turn the mine fan on high speed and repeat the step

Instructions and Equipment

The outline of the Brattice Window Method for determining the amount of leakage through a mine stopping, which is included in the lab handouts, will be followed as closely as possible in this experiment.

Once the Test Temporary Stopping has been put in place and the test holes in the stopping have been cut and measured, the fan will be turned on low speed. Subsequently, after hole #2 is taped shut, the air velocity through hole #1 will be measured. Once this has been accomplished, hole #2 will be opened (both will now be open) and velocity readings through #2 will be recorded.

Finally, the velocity of the air through hole #1 will be measured while hole #2 remains open. After the aforementioned tests are completed, the fan will then be turned on high speed and the procedure repeated.

Report

The report will consist of calculations relating to the determination of air leakage through the test temporary stopping using the Brattice Window Method. Included in the discussion should be an analysis of the difference in leakage for the two fan speeds; an explanation of any deviation from expected results; the various problems and advantages of the Brattice Window Method; and finally, the reasons why leakage determination for a mine's stoppings is so important.

Additional Information

Air Leakage Underground

Unfortunately, a lack of empirical data and knowledge of the condition of individual stoppings makes an exact analysis of underground stopping leakage impossible. Generally, leakage is most severe through the old stoppings out-bye the circuit. These are also subjected to higher pressure differential than the newer in-bye stoppings. Therefore, the circuit air volume diminishes at a decreasing rate progressing from out-bye to in-bye the circuit.

The amount of leakage depends on the stopping material and pressure differential across the stopping, and can be expressed as

Q = å H$_n$, in cfm

where å is the leakage at 1" W.G. and n is a constant which varies with stopping materials. Values of å range from 5 cfm for a coated stopping to 1,400 cfm for stoppings built of hollow slag concrete blocks; and values of n range from 0.3 for a hollow-core, dry-wall blocks to 1.1 for a foam coated concrete stopping.

Effect of Leaky Stoppings

The main effect of leaky stoppings is to increase the pressure which, in turn, the power requirements for ventilating a particular area with a specified amount of air. It can be shown theoretically and proved experimentally that the pressure increase that results from presence of leaky stoppings is approximately directly proportional to the total leakage air through the stoppings expressed as a percentage of the air volume delivered to the area to be ventilated.

Example 5-1 is a numerical example showing the effect of a leaky stopping. Assuming mine resistance is 1.00 x 10^{-10} lb. min^2/ft^4 in all sections, then the total fan pressure for a mine with perfect stoppings is,

$$\begin{aligned} H_T &= H_{AB} + H_{BC} + H_{CD} + H_{DE} + H_{EF} + H_{FG} + H_{GH} \\ &= 1(0.5)^2 + 1(0.5)^2 + 1(0.5)^2 + 1(0.5)^2 + 1(0.5)^2 + 1(0.5)^2 + 1(0.5)^2 \\ &= 1.75 \text{ in. W.G.} \end{aligned}$$

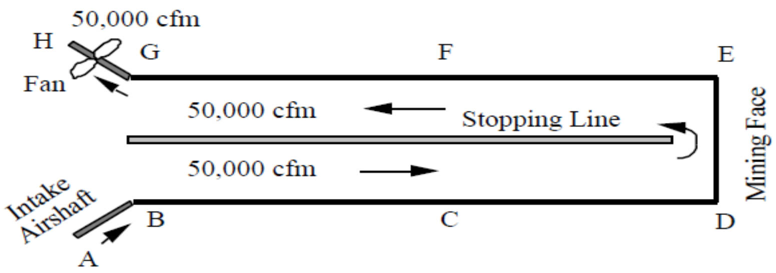

Figure 5-1 Mine panel with perfect stopping line

If 25,000 cfm of air leaks through Point I, an additional 25,000 cfm of air must enter the section to compensate for leakage (Figure 5-2).

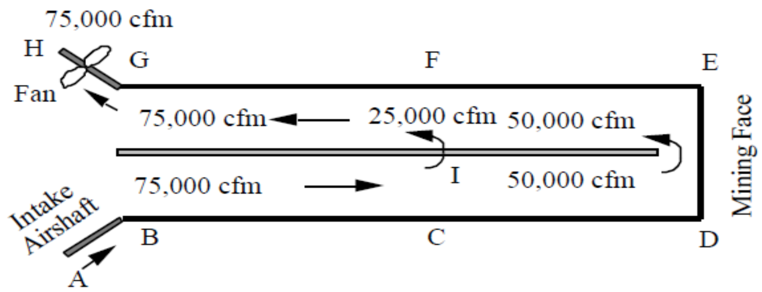

Figure 5-2 Mine panel with 25,000 cfm air leaking in the stopping line

In this case,

$$\begin{aligned} H_T &= H_{AB} + H_{BC} + H_{CD} + H_{DE} + H_{EF} + H_{FG} + H_{GH} \\ &= 1(0.75)^2 + 1(0.75)^2 + 1(0.5)^2 + 1(0.5)^2 + 1(0.5)^2 + 1(0.75)^2 + 1(0.75)^2 \\ &= 3.00 \text{ in. W.G.} \end{aligned}$$

Difference = 3.00 in. W.G. – 1.75 in. W.G. = 1.25 in. W.G.

Because the fan operates 24 hours a day, 365 days a year. For a fan having an 80% efficiency, 1.25 in. W.G. loss caused by the leaky stoppings with leakage of 25,000 cfm is equipment to:

$$\text{Brake Horsepower} = \frac{1.25 \text{ x } 25000}{6345 \text{ x } 0.8} = 6.156 \text{ hp}$$

or roughly \$2,414 per year, based on 6¢/kwh or 392.10/hp/year.

Averaging method.

The other method is to measure the average leakage through a group of stoppings as described by Stephens (2011):

This method is illustrated in Figure 4.3, which depicts a cut out of a coal mine section having 5 entries. Entries 1 and 2 have intake air, Entries 4 and 5 have return air, and Entry 3 has neutral air. The air courses are separated by stopping lines which periodically contain doors. Entry 3 has systematic box-check regulators to limit the neutral airflow quantity. The total intake flow is the combined measurements in Entries 1 and 2. If airflow measurements are made at section points A and B in the intake air course, the difference between A and B is the amount of intake air leaking through five stoppings into Entry 3. Pressure differentials between Entries 2 and 3 can easily be measured at the stoppings containing doors, yielding the average pressure differential across these five stoppings.

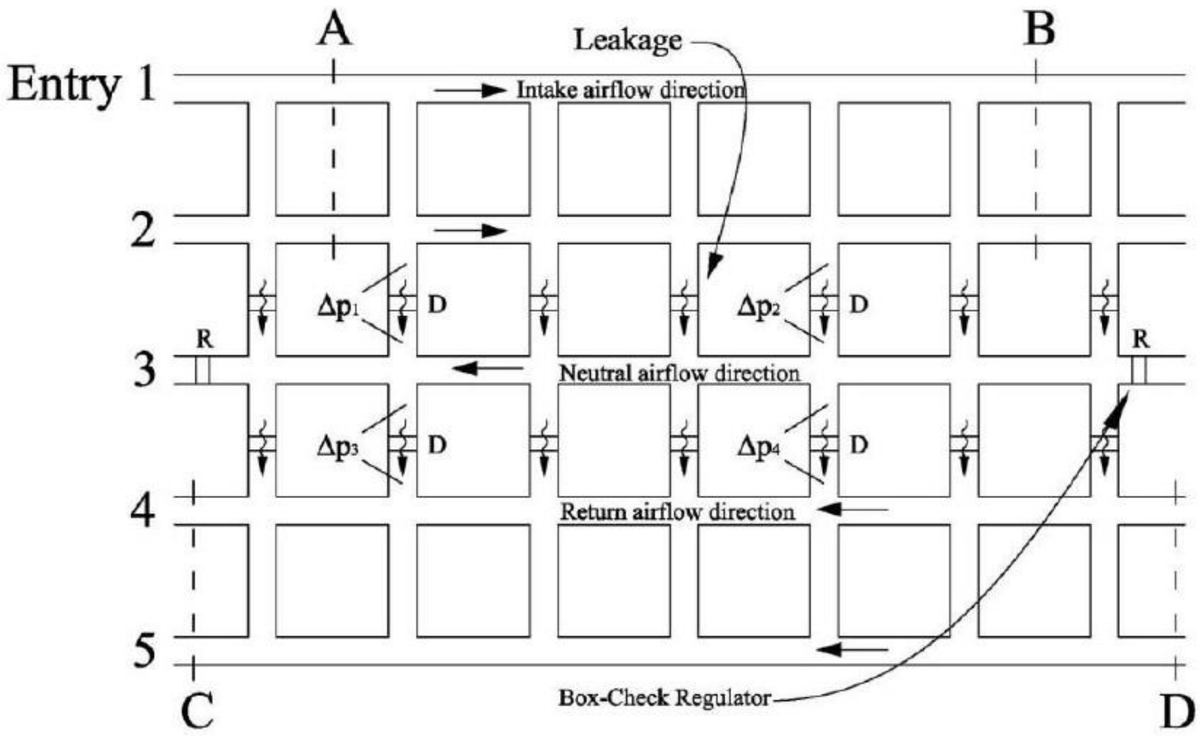

Figure 4.3 Standard method for measuring average stopping leakage

If the same method is used on the return side between points C and D the measurements yield the leakage through 7 stoppings. In both cases, Equation ($\Delta p = RQ^2$) can be used to quantify an equivalent resistance for the number of stoppings, and the individual stopping resistance is calculated using Equation $Re = R_i / N_a^2$ where Na is 5 for the intake side stoppings and 7 for the return side.

Square law of mine ventilation = $\Delta p = RQ^2$ ……………….….. (4.7)

Where, R = airway resistance (Ns^2/m^8)

Equivalent resistance $Re = R_i / N_a^2$ ………………………........ (4.8)

Where, R_i = resistance of a single airway (Ns^2/m^8), N_a = the number of parallel airways

Using this method assumes that the leakage quantities through all the stoppings in a particular group Qi, are equal (Qi = Q / Na) which is not necessarily the case.

For example, stoppings with doors generally allow greater leakage than those without. This assumption may be valid over short intervals where the differential pressure across the stoppings do not vary significantly, but it is not justified over longer intervals with high variance in differential pressures. This method is not intended to distinguish resistances between individual stoppings, but rather to determine an average resistance for a group of stoppings. The preferred interval is different for each mine and depends on the pillar dimensions, as well as stopping characteristics. This method provides the ability to measure resistance throughout a large mine relatively quickly while still being able to distinguish between groups of stoppings differing in age, condition, and type.

Airflow difference before and after stopping.

This method was used in this study to measure leakage across a stopping. It relies on the difference in air quantity measured before and after a stopping and neglects other losses due to resistance of shock. A simple mathematical equation 4.9 summarizes the whole concept:

$$Q_b - Q_a = \Delta Q \quad \text{.....................................}(4.9)$$

Where, Q_b = Quantity measured before stopping (m^3/s)

Q_a = Quantity measured after stopping (m^3/s)

ΔQ = Quantity of leakage (m^3/s)

Equation 4.9 is based on Kirchhoff's first law, meaning that the quantity measured before the stopping should be equal to the quantity measured after the stopping. The distance before or after the stopping where measurements are taken is not standard but rather expected to be not less than 1.5 meters.

Leakage at Missouri S&T Experimental Mine.

Air quantity measurements were taken at the Missouri S&T Experimental Mine using the method described as above to measure the quantity of leakage. These measurements were taken at individual stoppings throughout the mine, including in the return airways.
A calibrated anemometer was used to measure the airflow velocity and a tape measure used to measure the airway dimensions. The map in Figure 4.4 shows the stoppings at which leakage measurements were taken. In order to draw a fairly reasonable and in depth conclusion, a number of different scenarios were experimented on. These included either the main fan running alone or coupled with one of the booster fans or both of the booster fans. Also the booster fan blade angles were varied to have a sizable difference in the amount of pressure from the booster fan. The main fan speed was maintained to a maximum at all times throughout the experiment since the emphasis of the study was on booster fans. Varying the speed of the main fan would introduce an unwanted variable in the data. Leakage through the Kennedy machine doors was not taken into consideration.
For all the scenarios tested, the integrity of the stoppings was maintained to the highest possible standard. The stoppings were sealed for any excessive leakage after every test run and every booster fan blade angle change.

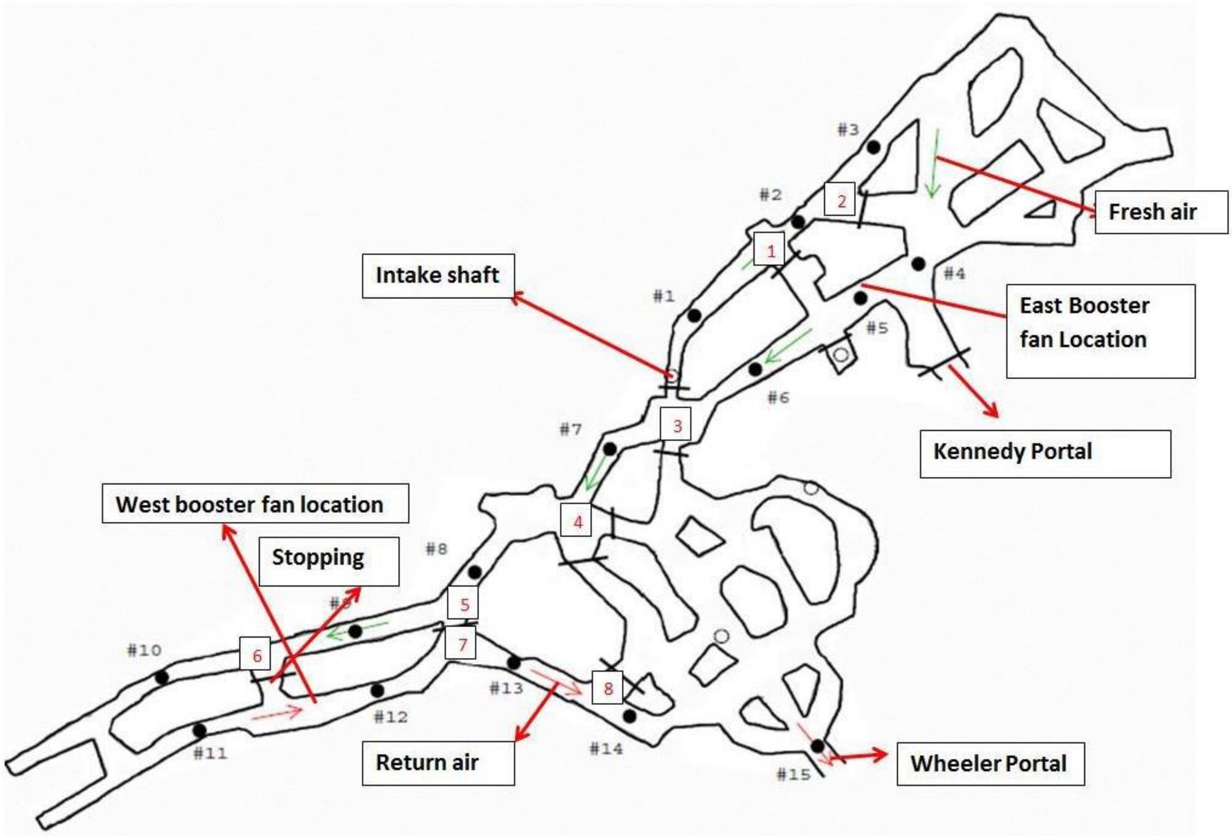

Figure 4.4 Map showing stoppings where leakage was measured

RECIRCULATION

Recirculation is a form of reusing air to ventilate an airway as the air passes the same district more than once. Jones (1986) defined recirculation as the movement of mine ventilation air past the same point more than once. Recirculation occurs when air leaks from the return airways into the intake airways as a result of high pressures in the return airways than that in the intake airways. Recirculation in a ventilation circuit occurs in two forms: controlled and uncontrolled recirculation. Controlled recirculation is a term that is used to describe a recirculation circuit that is purposefully designed and utilized in a controlled fashion to provide some ventilation benefits without adversely affecting other ventilation variables (Hartman et al., 1997), fans are placed in a mine ventilation circuit to produce a desired recirculation quantity. In controlled recirculation systems, a portion of the return air is purposefully directed into the intake air and transmitted to the production areas and the quantity of recirculated air is closely monitored and managed (Calizaya, 2009).

The use of controlled recirculation circuits is considered to be beneficial in mines where;

- Mine intake air must be heated because of cold climates.
- Mine air is refrigerated for reasons of comfort or productivity.
- Added velocity at the face would result in better turbulent mixing of air and methane at the point of release.
- Added velocity at the face would more effectively carry away dusts.
- Working faces are far removed from the mine portals, such as in under sea mining.

Uncontrolled recirculation is an unplanned and unexpected air leakage from the return airways into the intake airways. This type of recirculation is not managed and therefore has been a deterrent in the use of booster fans in United States underground coal mines. The recirculation of mine air has in the past been avoided principally because of the fear of a build-up of pollutants, particularly methane concentration in the general body of the air. Controlled recirculation has been a known form of ventilation for some time.

CONCLUSION

From the investigations of behaviour in recirculation and leakage under the influence of booster fans, these conclusions can be drawn;

- The amount of leakage created does not depend on the booster fan used but rather on the integrity of the stopping. This is shown by stoppings that have higher leakage rates at lower blade angle settings and lower pressure differentials than those with higher values. The highest leakage rates are observed when both booster fans are in use and at blade angle settings 4 and 5.
- Using the booster fan creates higher pressure at the part of the mine where the booster fan is located. This in turn creates higher leakage rates at the stoppings furthest from the booster fan. The west booster fan creates the highest leakage rate at the eastern part of the mine while the east booster fan creates the highest leakage rates at the western part of the mine. When using both booster fans, the higher leakage rates are spread unevenly throughout the stoppings in the mine.
- Recirculation observed at stopping 2 is not a result of the influence of the use of booster fans. This is because all fan combinations and the mine main fan only both experience such recirculation.
- Recirculation occurs when blade angle settings 4 and 5 are used for all the fan combinations which mean that they are oversized for the mine. The ideal blade setting would be blade angle setting 3 since it is not oversized or undersized for the mine.
- Recirculation is most pronounced at the stoppings that are closest to the booster fan that is being used. The recirculation that is caused by the east booster fan is concentrated at stopping 1 while that caused by the west booster fan is concentrated at stoppings 4 and 5.