Ventilasi Tambang
Kuliah 11

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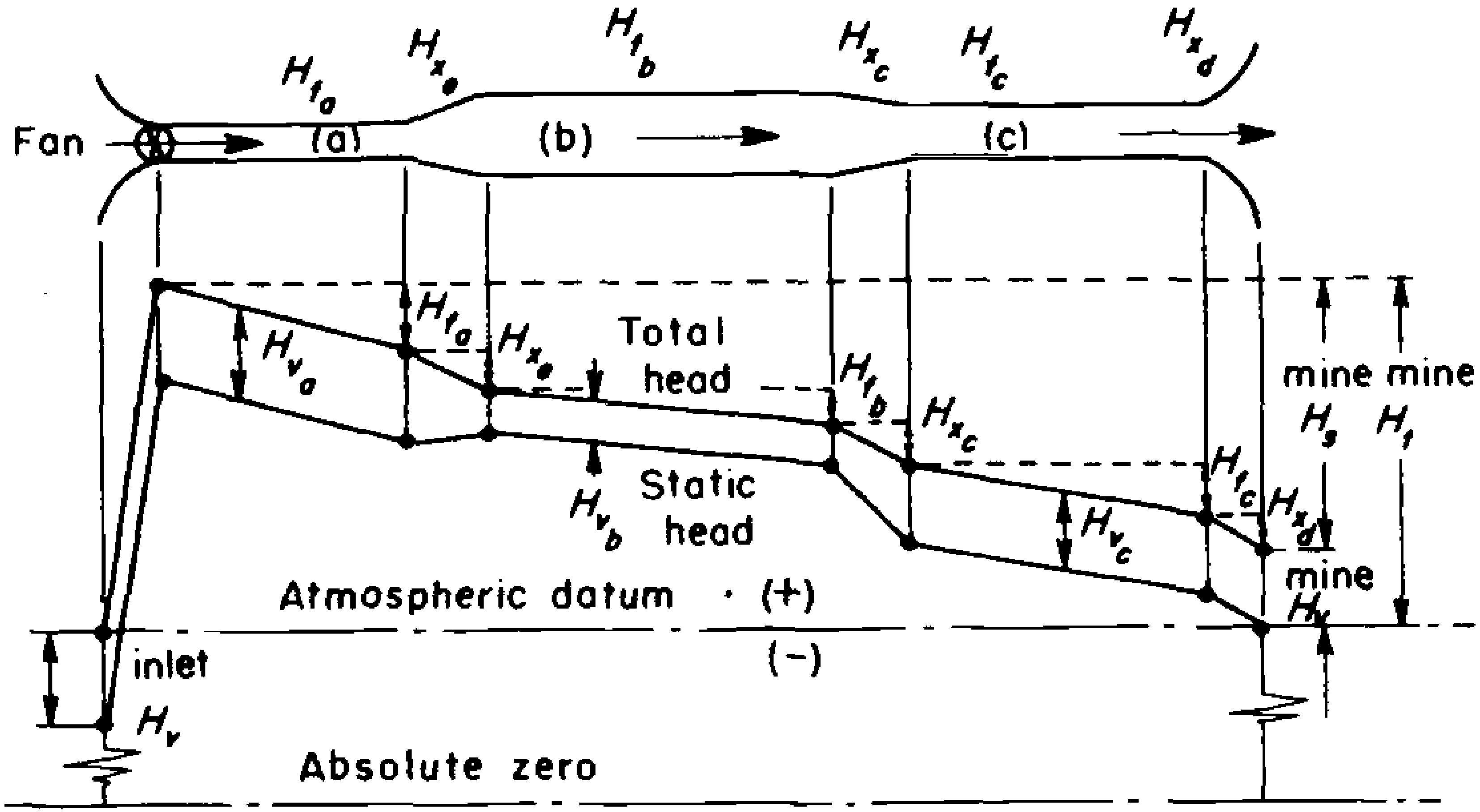
Fakultas Teknik Pertambangan dan Perminyakan - ITB

Perencanaan Ventilasi Tambang - 1

TA3121 – Ventilasi Tambang

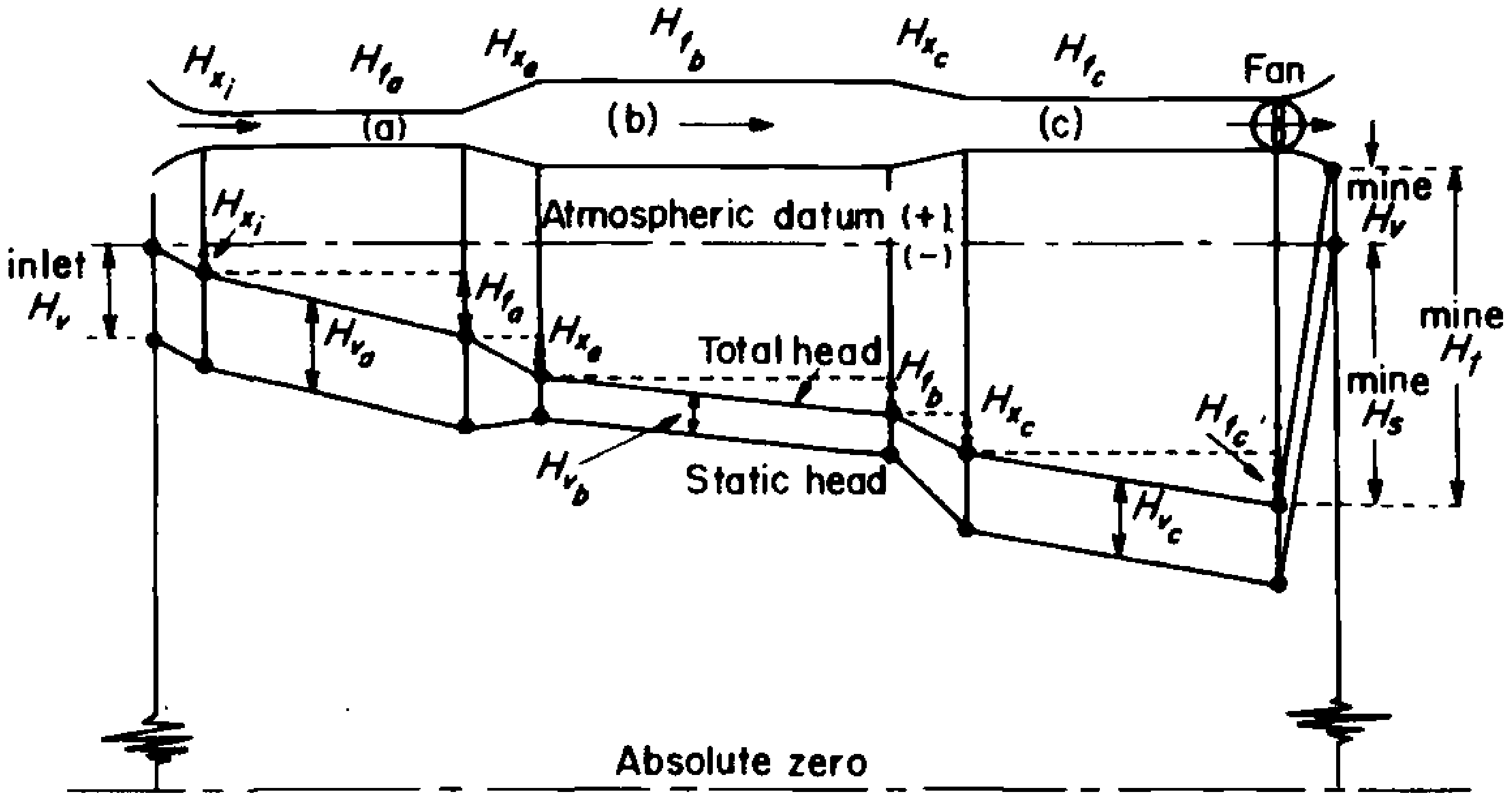
HAK CIPTA

GRADIEN KEHILANGAN TEKANAN PADA JALUR VENTILASI SISTEM HEMBUS



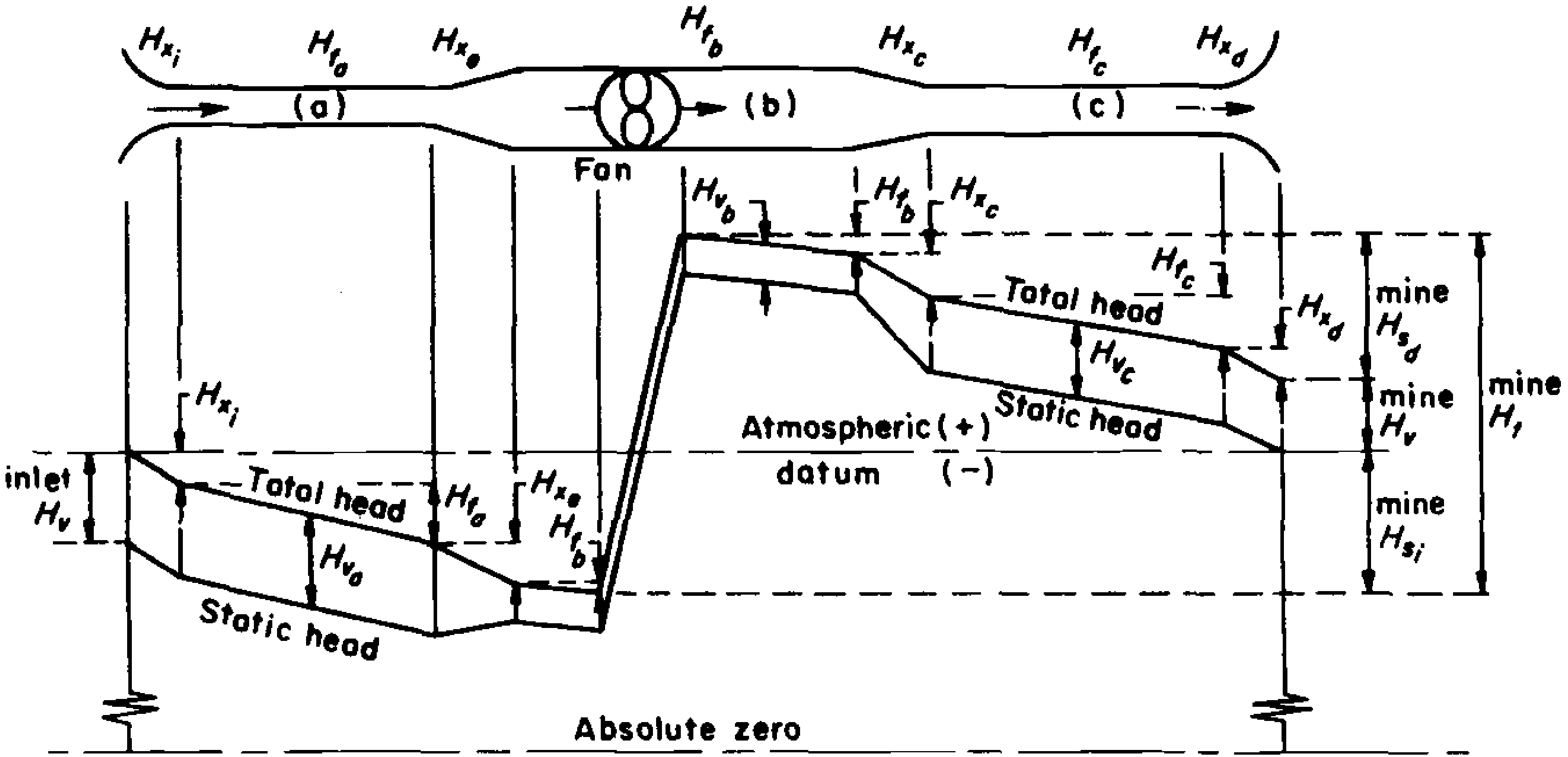
Hartman *dkk.* (1997)

GRADIEN KEHILANGAN TEKANAN PADA JALUR VENTILASI SISTEM HISAP



Hartman *dkk.* (1997)

GRADIEN KEHILANGAN TEKANAN PADA JALUR VENTILASI SISTEM BLOWER



Hartman *dkk.* (1997)

PERHITUNGAN KURVA KARAKTERISTIK TAMBANG

$$R = kL\frac{per}{A^3} \qquad \frac{Ns^2}{m^8} \text{ or } \frac{kg}{m^7}$$

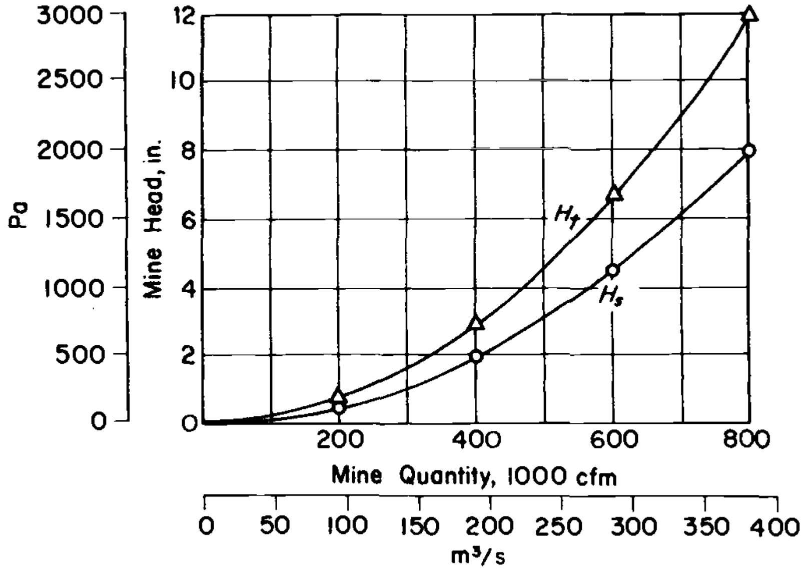
$$p = RQ^2 \qquad \text{Pa}$$

$H_1=RQ_1^2$; $H_2=RQ_2^2$ → substitusi R → $\frac{H_1}{H_2} = \left(\frac{Q_1}{Q_2}\right)^2$

Example 7.1 Given a mine with a single fan whose static head is 2 in. water (497.7 Pa) and total head 3 in. water (746.5 Pa) at a quantity of 400,000 cfm (188.8 m^3/s), determine and plot the mine characteristic curve.

Solution: Assume quantities and calculate the corresponding heads by Eq. 7.6.

Mine Q cfm (m^3/s)	Mine H_s in. water (Pa)	Mine H_t in. water (Pa)
0 (0)	0 (0)	0 (0)
200,000 (94.4)	0.5 (125)	0.8 (200)
400,000 (188.8)	2.0 (497)	3.0 (745)
600,000 (283.2)	4.5 (1120)	6.8 (1690)
800,000 (377.6)	8.0 (1990)	12.0 (2990)



Hartman *dkk.* (1997)

CONTOH PERHITUNGAN KEHILANGAN TEKANAN PADA JALUR VENTILASI

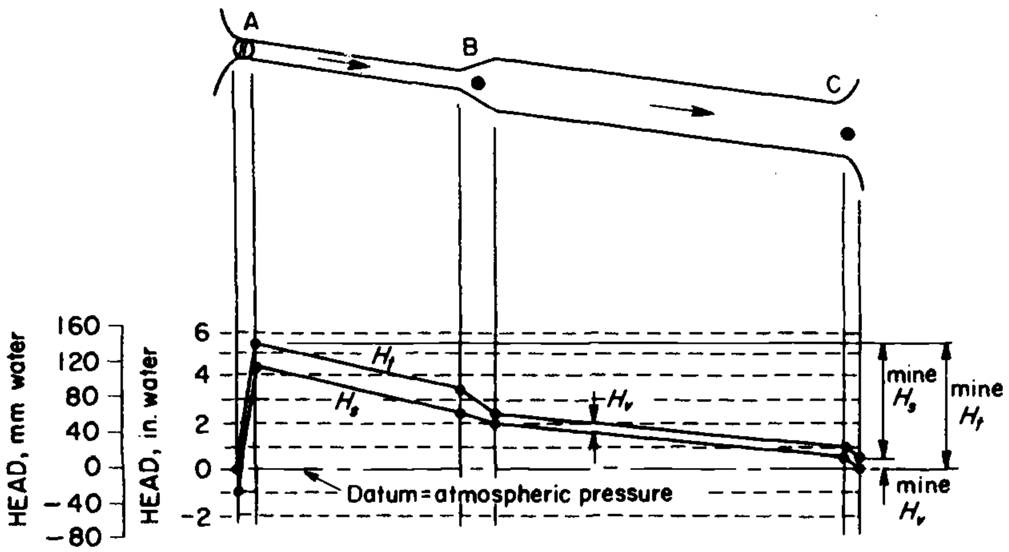


FIGURE 5.6 Head gradients for the blower system described in Example 5.2

For duct AB:	Friction loss = 2.0 in. (50.8 mm) water
	Velocity head = 1.0 in. (25.4 mm)
For duct BC:	Friction loss = 1.5 in. (38.1 mm)
	Velocity head = 0.5 in. (12.7 mm)
For inlet A:	Shock loss = 1.0 in. (25.4 mm), without fan
For expansion B:	Shock loss = 1.0 in. (25.4 mm)
For discharge C:	Shock loss = 0.5 in. (12.7 mm)

Assume the velocity head at discharge equals that in duct BC. Also calculate and show the mine heads.

Solution: See Fig. 5.6. Start plotting gradients from discharge at C, working backward to fan at A. Shock loss due to inlet can be disregarded. Values of mine heads calculated by Eqs. 5.7 and 5.8 are

$$\text{Mine } H_s = H_{f_{AB}} + H_{f_{BC}} + H_{x_A} + H_{x_B} + H_{x_C}$$

$$= 2.0 + 1.5 + 0 + 1.0 + 0.5$$

$$= 5.0 \text{ in. (127.0 mm) water}$$

$$\text{Mine } H_v = H_{v_{BC}} = 0.5 \text{ in. (12.7 mm)}$$

$$\text{Mine } H_t = 5.0 + 0.5 = 5.5 \text{ in. (139.7 mm)}$$

Hartman *dkk.* (1997)

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A Review of Primary Mine Ventilation System Optimization

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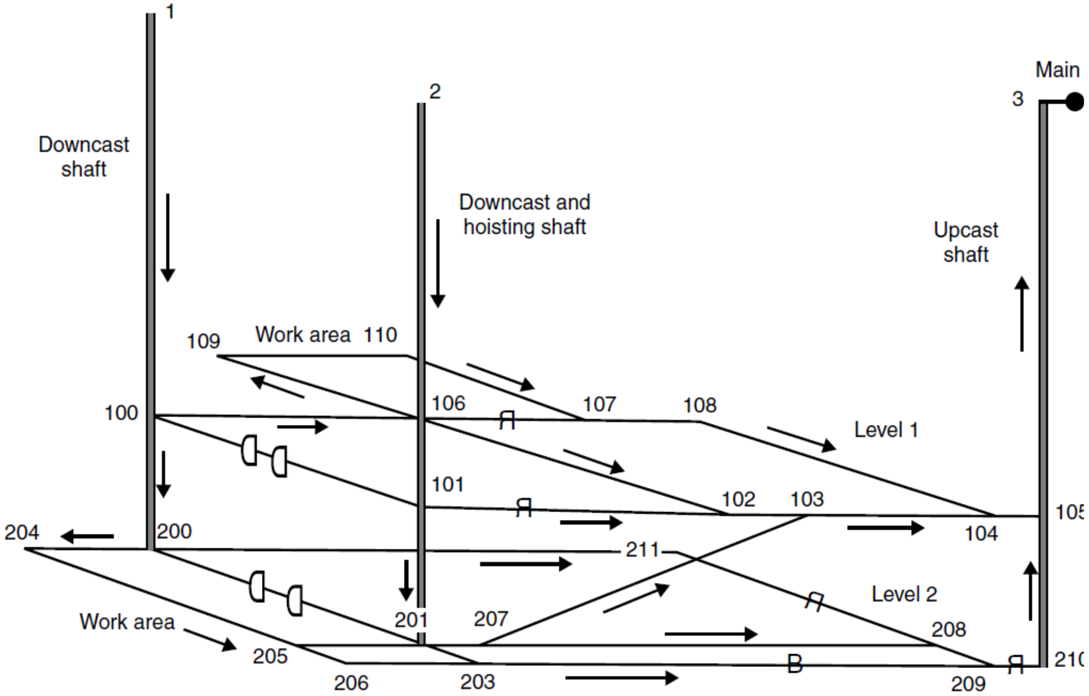


Figure 1: This example of a generic mine ventilation network shows the direction of airflow and typical locations of the ventilation devices, such as doors (DD), main fan, booster fan (B), and regulators (R), which maintain the required airflow distribution.
Source. McPherson (1993).

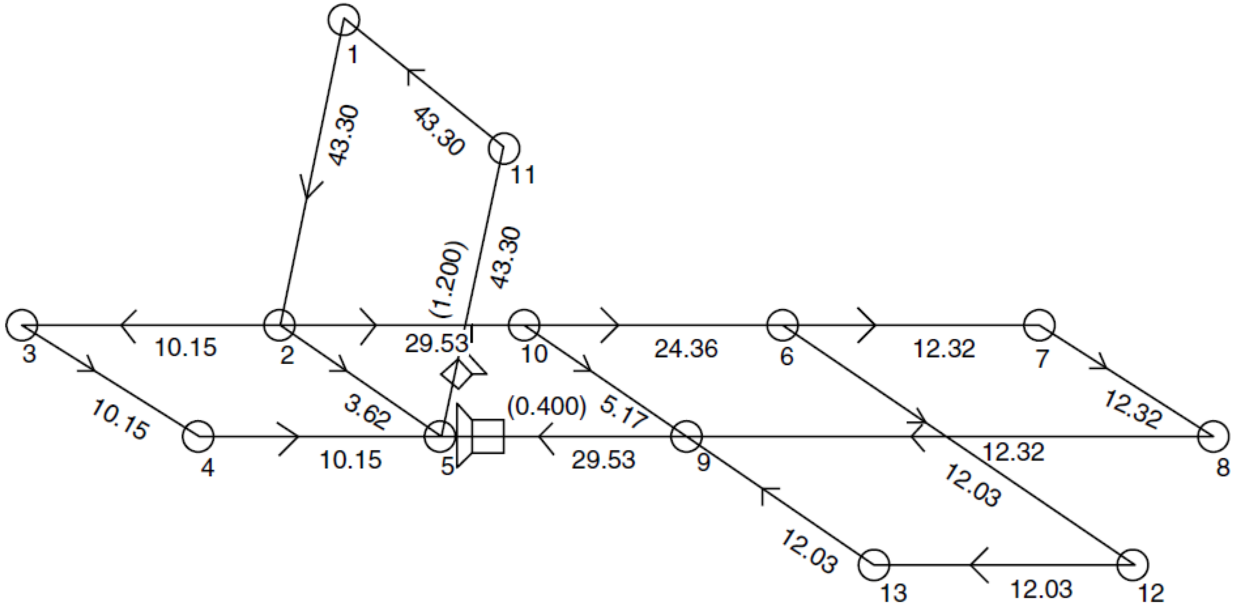
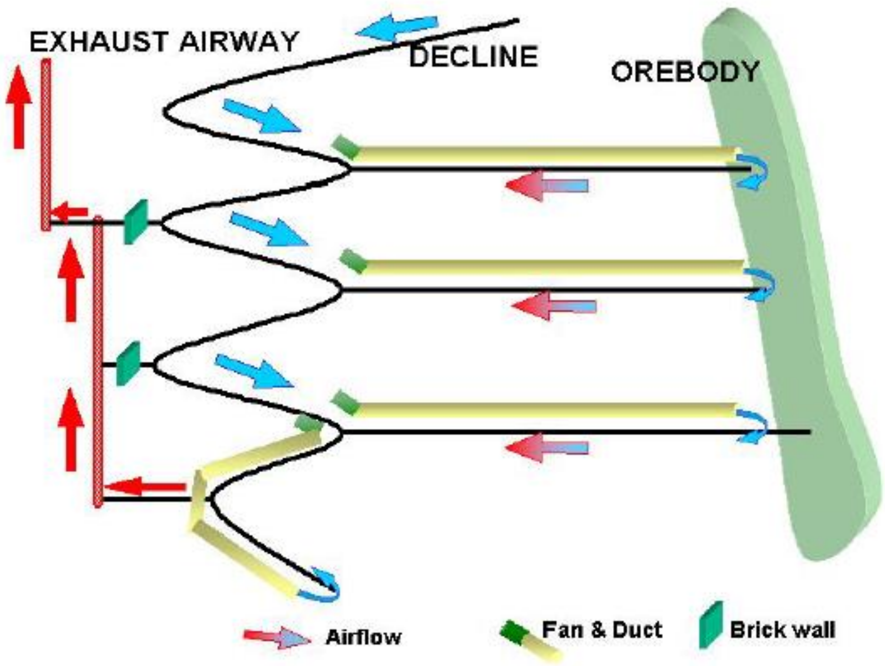


Figure 3: This schematic shows a simple small-scale mine ventilation network.
Source. Calizaya et al. (1987).

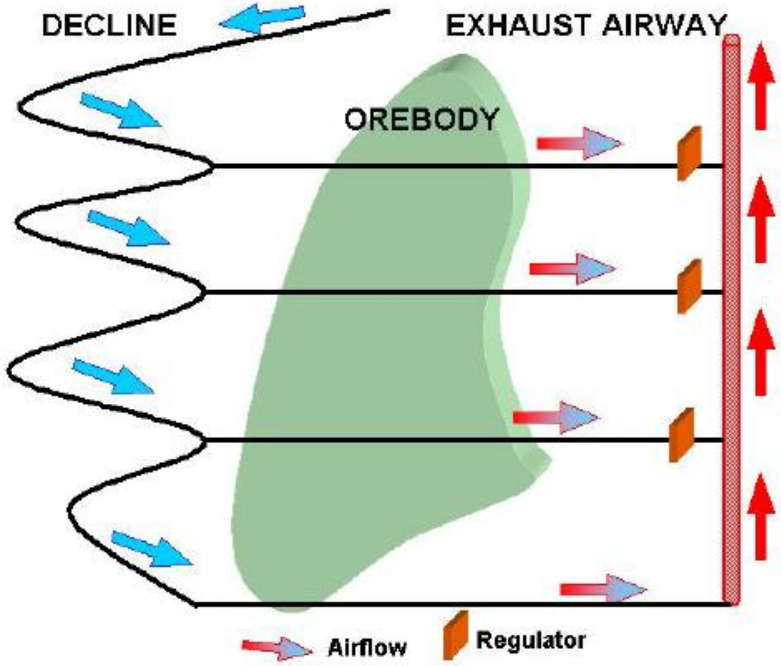
Acuna & Lowndes (2014)

SISTEM VENTILASI KESELURUHAN (TAMBANG MINERAL)

Parallel systems exist in mines when air is exhausted on a number of levels.



series



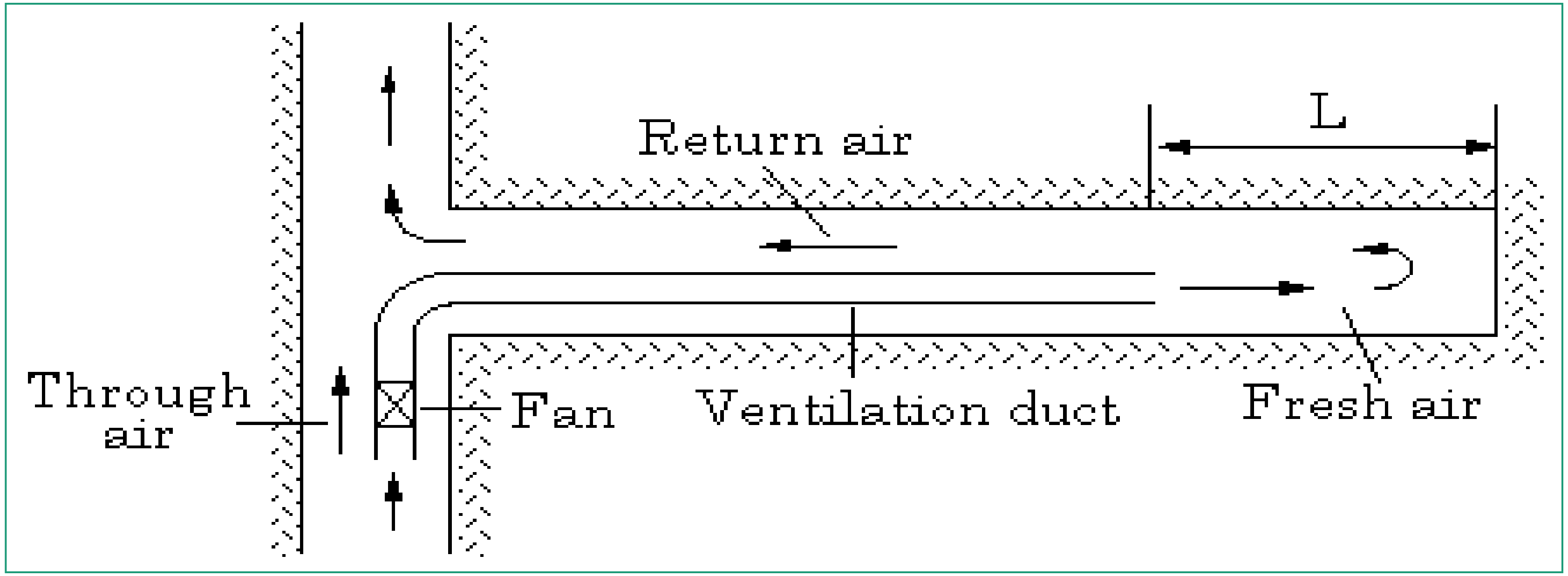
paralel

Source: Course literature

SISTEM VENTILASI LOKAL

Sistem Hembus Sederhana (*Simple Forcing*)

- udara bersih dihembuskan ke permuka kerja melalui pipa udara *(duct)* dengan kecepatan tertentu, dan udara kotor dari permuka kerja akan mengalir melalui terowongan (*tunnel)* tersebut.



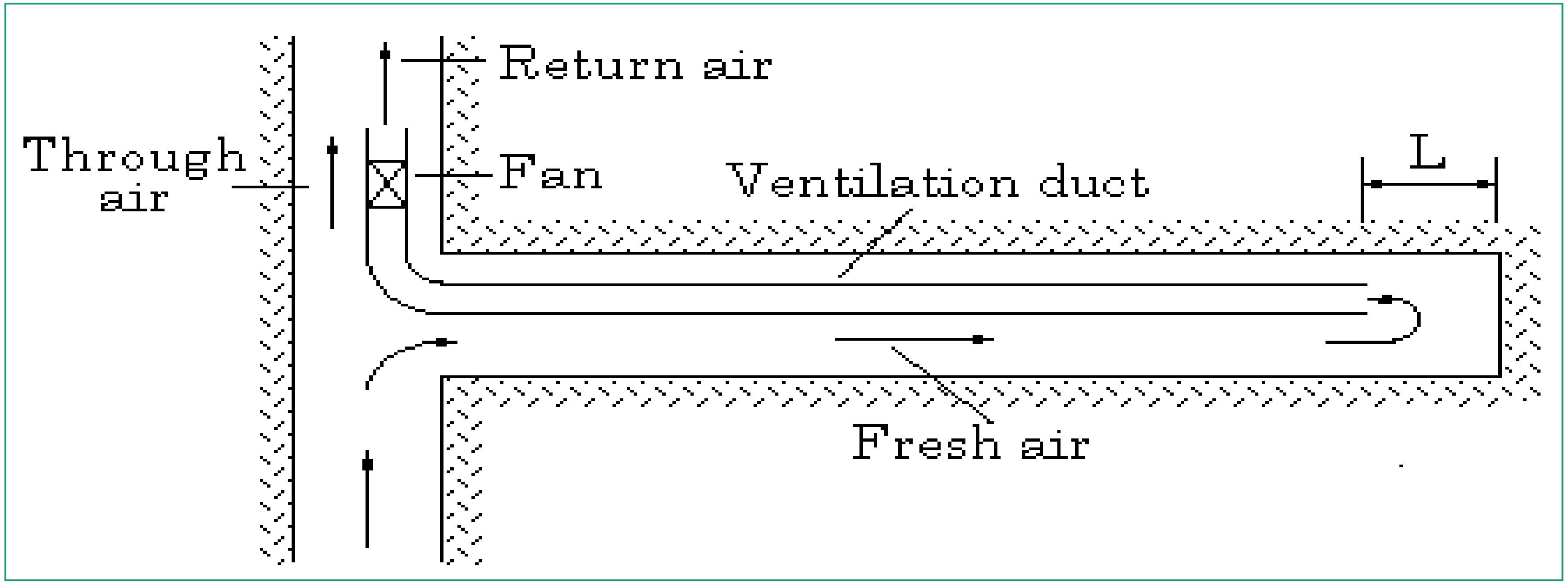
Sistem Hembus Sederhana (*Simple Forcing*)

Source: Course literature

SISTEM VENTILASI LOKAL

Sistem Isap Sederhana (*Simple Exhaust*)

- udara kotor dari permuka kerja di isap oleh kipas angin tambahan. Sehingga udara bersih akan mengalir ke permuka kerja melalui terowongan *(tunnel)*.



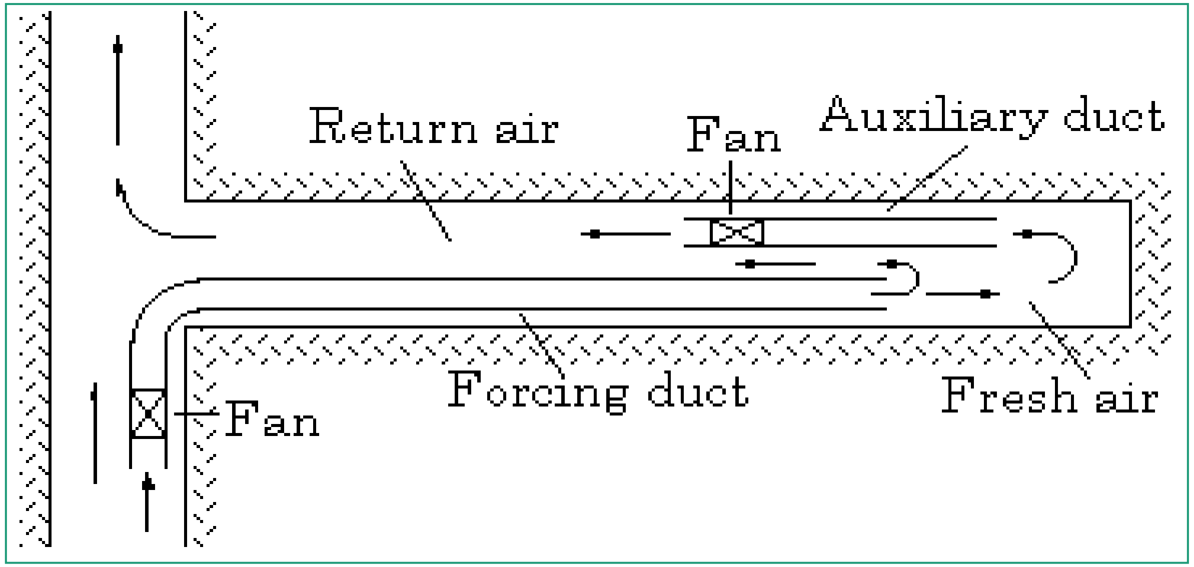
Sistem Isap Sederhana (*Simple Forcing*)

Source: Course literature

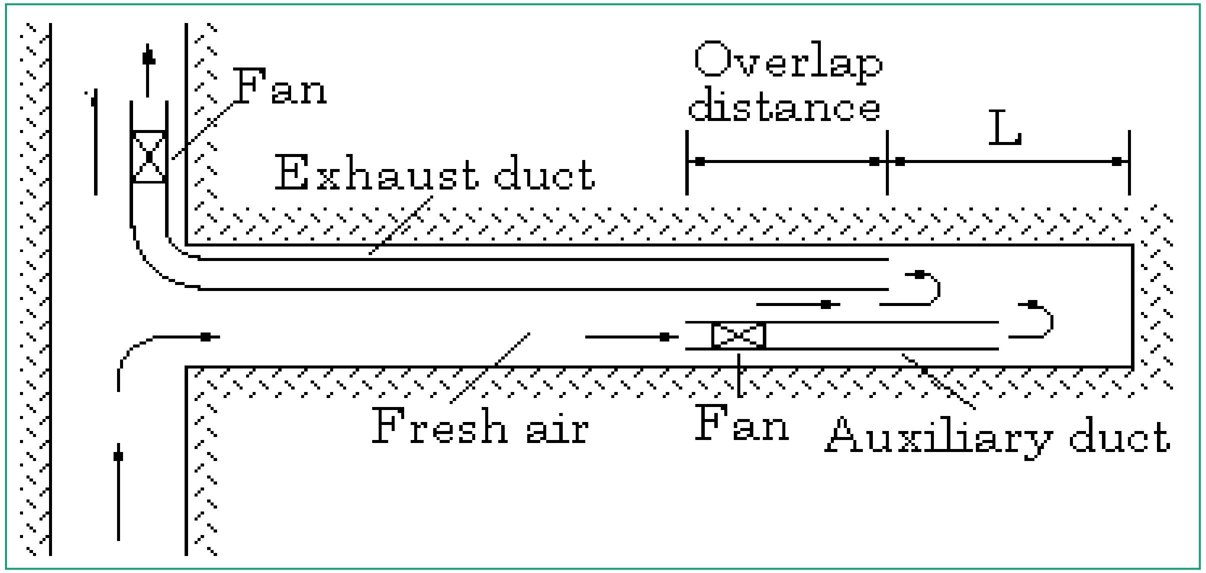
SISTEM VENTILASI LOKAL

Sistem Kombinasi Hembus dan Isap (*Overlap System*)

- udara bersih dihembuskan ke permuka kerja dan udara kotor yang berasal dari kegiatan dipermuka kerja diisap oleh kipas angin tambahan yang biasanya dilengkapi dengan “*dust collector*”.



Forcing With Exhaust Overlap



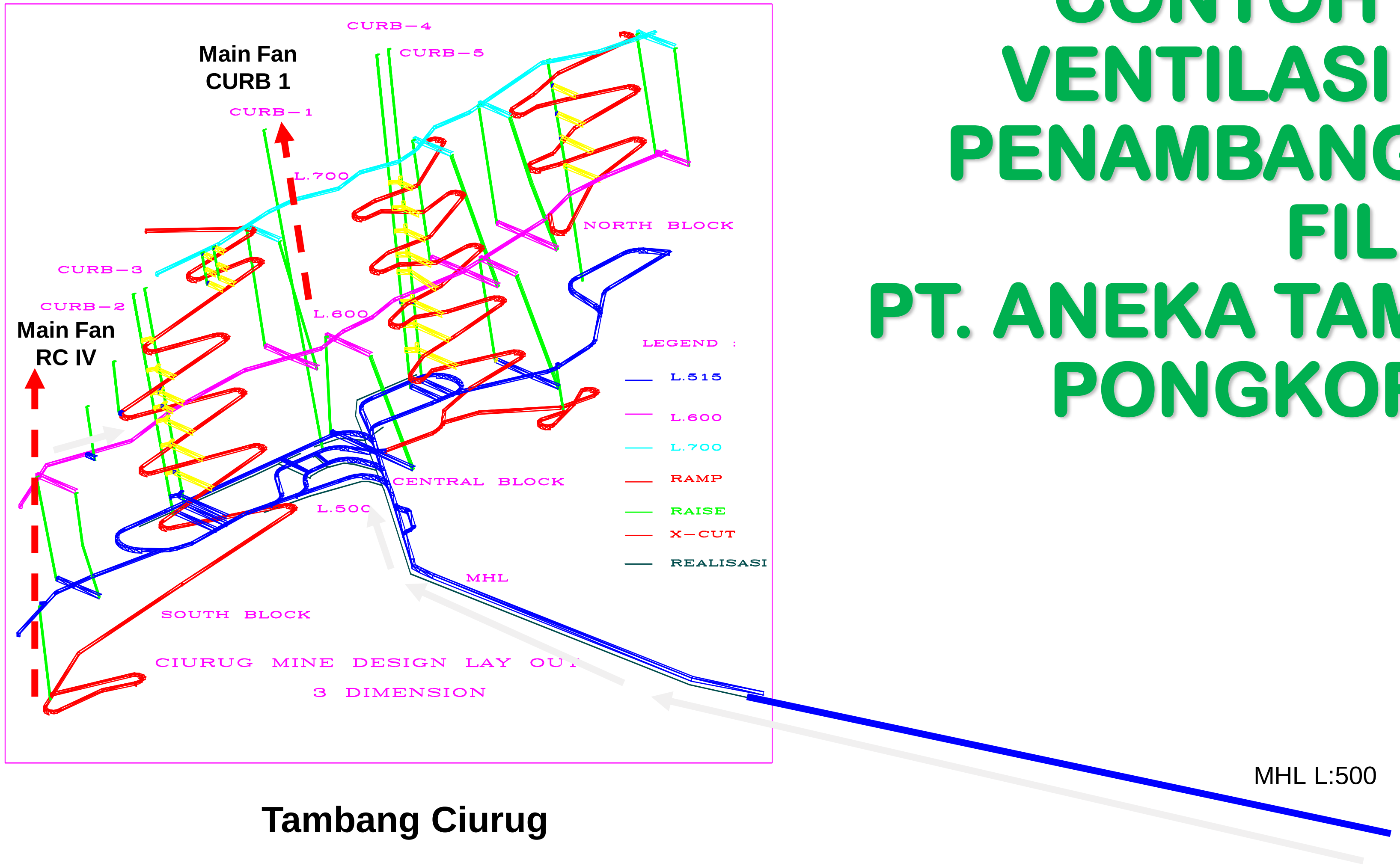
Exhaust With Forcing Overlap

Source: Course literature

PIPA UDARA (*DUCT*)

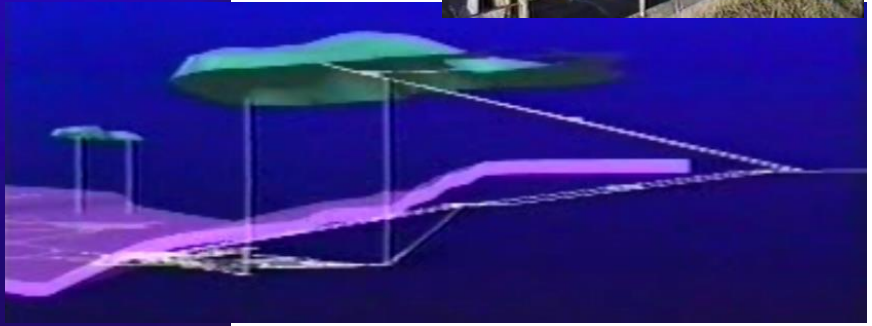
- Pipa udara (*Duct*) biasanya digantung pada langit-langit terowongan.
- Diusahakan tidak bocor, tidak terjadi tekukan-tekukan sehingga udara dapat disalurkan ke permuka kerja dengan efisien.
- Untuk pipa udara sedapat mungkin berukuran besar sehingga mempunyai tahanan yang kecil.
- Tetapi harus diperhitungkan pula jarak aman antara tinggi peralatan mekanis yang digunakan dalam tambang dengan *duct* untuk menghindari terjadinya kerusakan.
- *Duct* biasanya tidak digunakan untuk keseluruhan sistem ventilasi tambang karena luas penampang yang kecil (dibandingkan dengan jalur utama) sehingga akan menyebabkan terjadi tahanan yang sangat besar.
- Bahan dari *vent duct* adalah bahan *flame retardant* (will not sustain burning)

CONTOH SISTEM VENTILASI METODE PENAMBANGAN CUT & FILL PT. ANEKA TAMBANG, TBK. PONGKOR (2006)

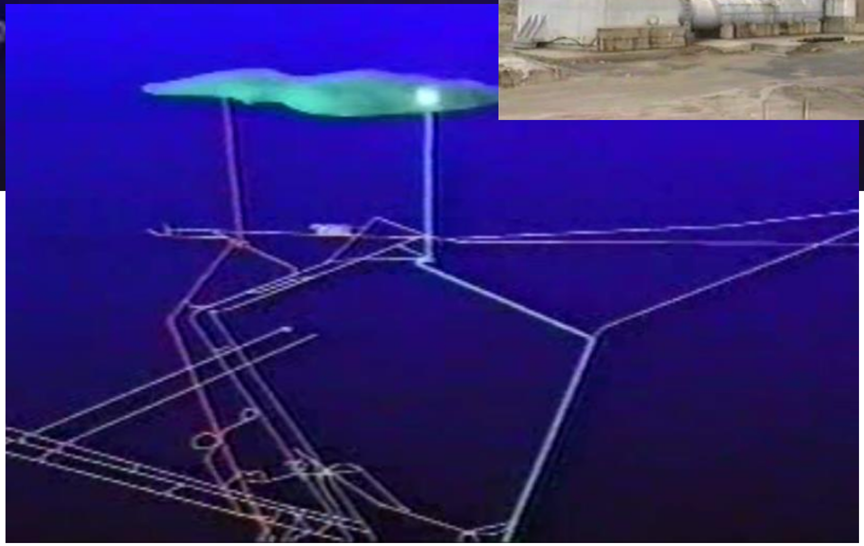


Tambang Ciurug

CONTOH SISTEM VENTILASI TAMBANG BATUBARA LONGWALL IKESHIMA, JEPANG



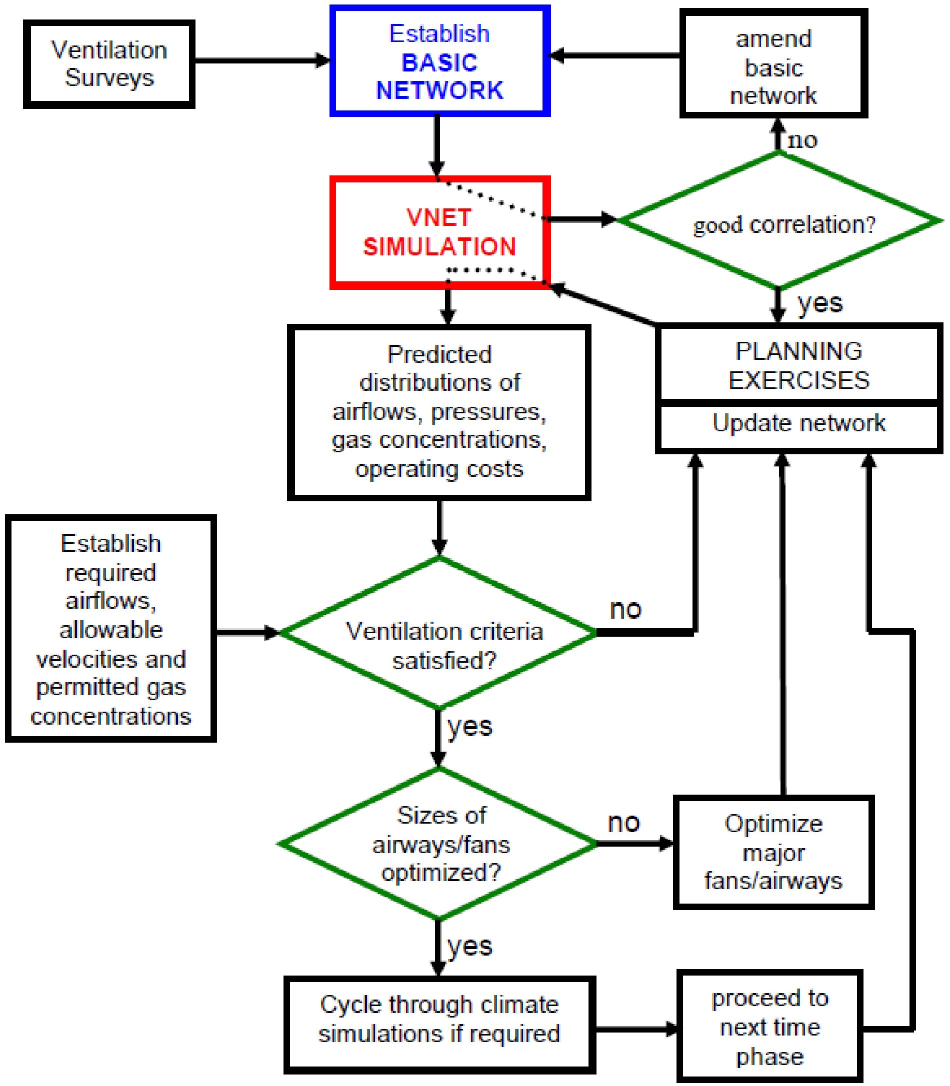
IKESHIMA
2 jalur udara vertikal
2 jalur udara miring



HIKISHIMA
2 jalur udara vertikal

Video Pelatihan Ventilasi
JCOAL (2008)

DIAGRAM ALIR PERENCANAAN VENTILASI TAMBANG



McPherson (1993)

KEBUTUHAN UDARA TAMBANG

Jumlah Pekerja di Dalam Tambang Bawah Tanah

- volume udara bersih yang dialirkan dalam sistem ventilasi harus diperhitungkan berdasarkan jumlah pekerja terbanyak pada suatu lokasi kerja dengan ketentuan untuk setiap orang tidak kurang dari 2 meter kubik per menit selama pekerjaan berlangsung.

Jumlah Peralatan Tambang yang Bekerja

- volume udara bersih yang dialirkan dalam sistem ventilasi harus ditambah sebanyak 3 meter kubik per menit untuk setiap tenaga kuda, apabila mesin diesel dioperasikan.

Ruangan Kerja yang Terdapat di dalam Tambang Bawah Tanah

- Untuk memperkirakan kebutuhan udara pada lokasi workshop di bawah tanah, perlu dibandingkan kebutuhan udara berdasarkan jumlah peralatan bermesin diesel dengan kebutuhan udara berdasarkan laju pergantian udara (misalnya 10 kali pergantian udara per jam) (McPherson, 1993). Perhitungan debit yang lebih besar yang dipergunakan sebagai acuan.

KEBUTUHAN UDARA TAMBANG

Jumlah Strata Gas

Untuk memperkirakan kebutuhan udara untuk mengatasi strata gas dapat dihitung dengan menggunakan persamaan.

$$Q = \frac{100\ Eg}{Cg}$$

(McPherson, 1993)

Keterangan:

Q = Debit yang dibutuhkan (m^3/detik)

E_g = Laju emisi gas (m^3/detik)

C_g = Konsentrasi umum gas di dalam udara yang diijinkan (dituju) (% volume)

- Besarnya Cg sering diambil sebesar setengah dari batas konsentrasi yang diijinkan oleh peraturan untuk dilakukan penanganan.

KEBUTUHAN UDARA TAMBANG

Jumlah Debu < 5 mikron (untuk mineral)

Untuk memperkirakan kebutuhan udara untuk mengatasi debu < 5 mikron (untuk mineral) dapat dihitung dengan menggunakan persamaan (McPherson, 1993)

$$Q = \frac{Ed}{Cd} x \frac{P}{3600}$$

Keterangan:

Q = Debit yang dibutuhkan (m^3/detik)

E_d = Laju emisi *respirable dust* (mg/ton)

P = Tingkat produksi mineral (ton/jam)

C_d = Kenaikan konsentrasi *respirable dust* di dalam udara yang diijinkan (dituju) (mg/m^3)

Catatan:

C_d = Ambang batas konsetrasi *respirable dust* – konsentrasi *respirable dust* dalam udara masuk.

KEBUTUHAN UDARA TAMBANG

Jumlah Gas Hasil Peledakan

- Untuk memperkirakan kebutuhan udara dalam mengencerkan gas hasil kegiatan peledakan, perlu dilakukan perkiraan jumlah gas berbahaya dan beracun yang muncul dari reaksi peledakan (De Souza, E.M., and Katsabanis, 1991). Perlu dipertimbangkan jumlah bahan peledak dan jenis reaksi (apakah *zero/negative/positive oxygen balance*).

TABLE 8
Volume, maximum allowable concentration and re-entry period for gases produced using 132 kg AN/FO

Gas	Volume per kg explosive (m^3)	Gas concentration	TLV	Re-entry period (min)
CO	0.001	0.000148	0.0001	0.9
CO_2	0.024	0.00355	0.005	0
NO	0.002	0.000296	0.000005	9.4
NO_2	0.0014	0.000207	0.000005	8.6
CH_4	0.00013	0.0000192	0.001	0

De Souza and Katsabanis (1991)

Temperatur dan Kelembaban

- Temperatur udara di dalam tambang bawah tanah harus dipertahankan antara 18 derajat Celcius sampai dengan 27 derajat Celcius dengan kelembaban relatif maksimum 85 persen.

KRITERIA KECEPATAN UDARA MAKSIMUM TAMBANG BAWAH TANAH (McPHERSON, 1993)

Jenis Jalur Udara	Kecepatan maksimum (m/s)
Permuka kerja	4
Conveyor drifts	5
Main haulage routes	6
Smooth lined main airways	8
Hoisting shafts	10
Ventilation shafts	20

CONTOH KEGIATAN DI TAMBANG (PT ANTAM)

Aktivitas pekerja di tambang bawah tanah

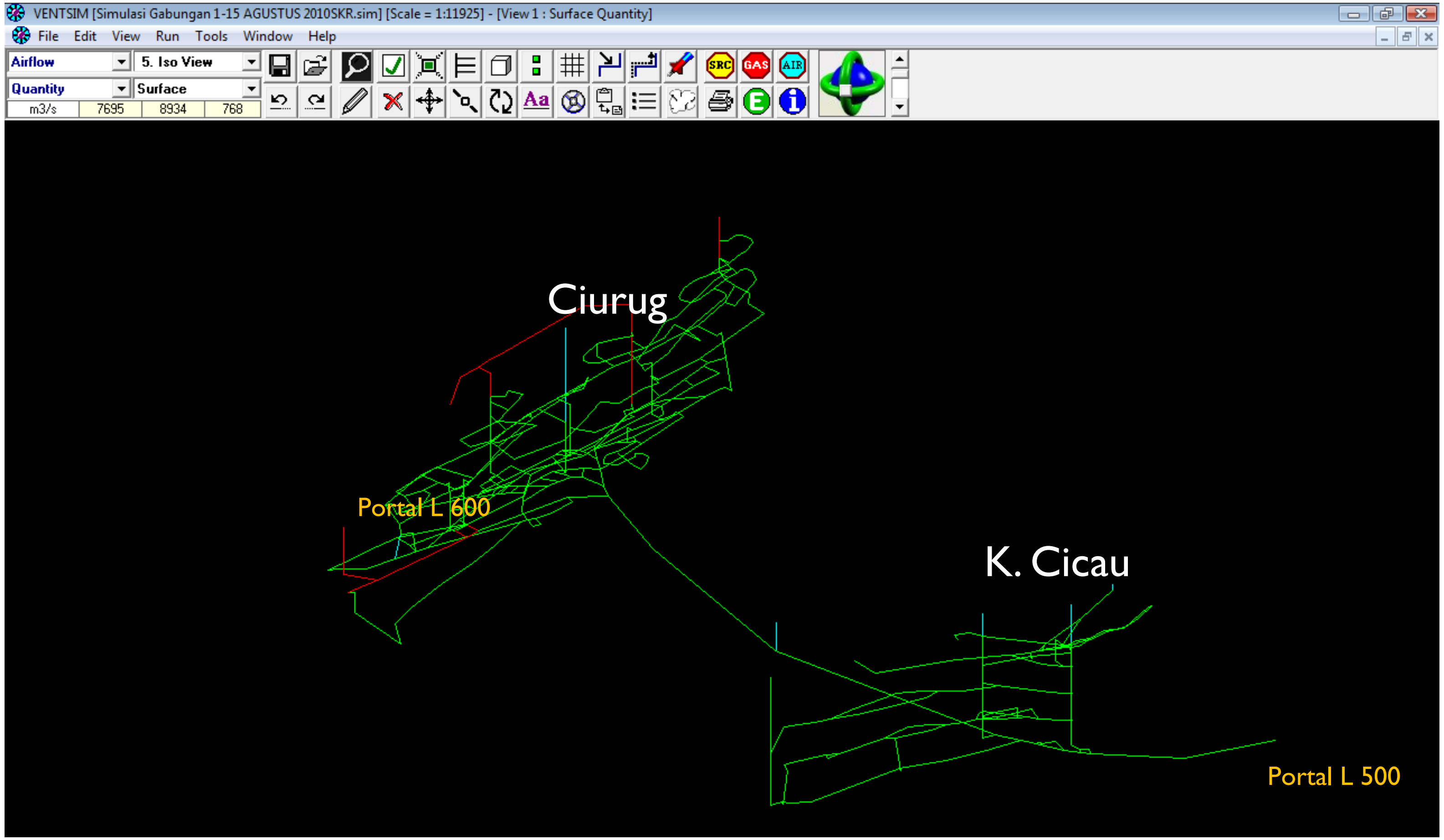
Pengeboran dengan Jumbo Drill

Pemasangan penyangga dengan bantuan Wheel Loader (56 kW)

LHD (120 kW) di workshop bawah tanah

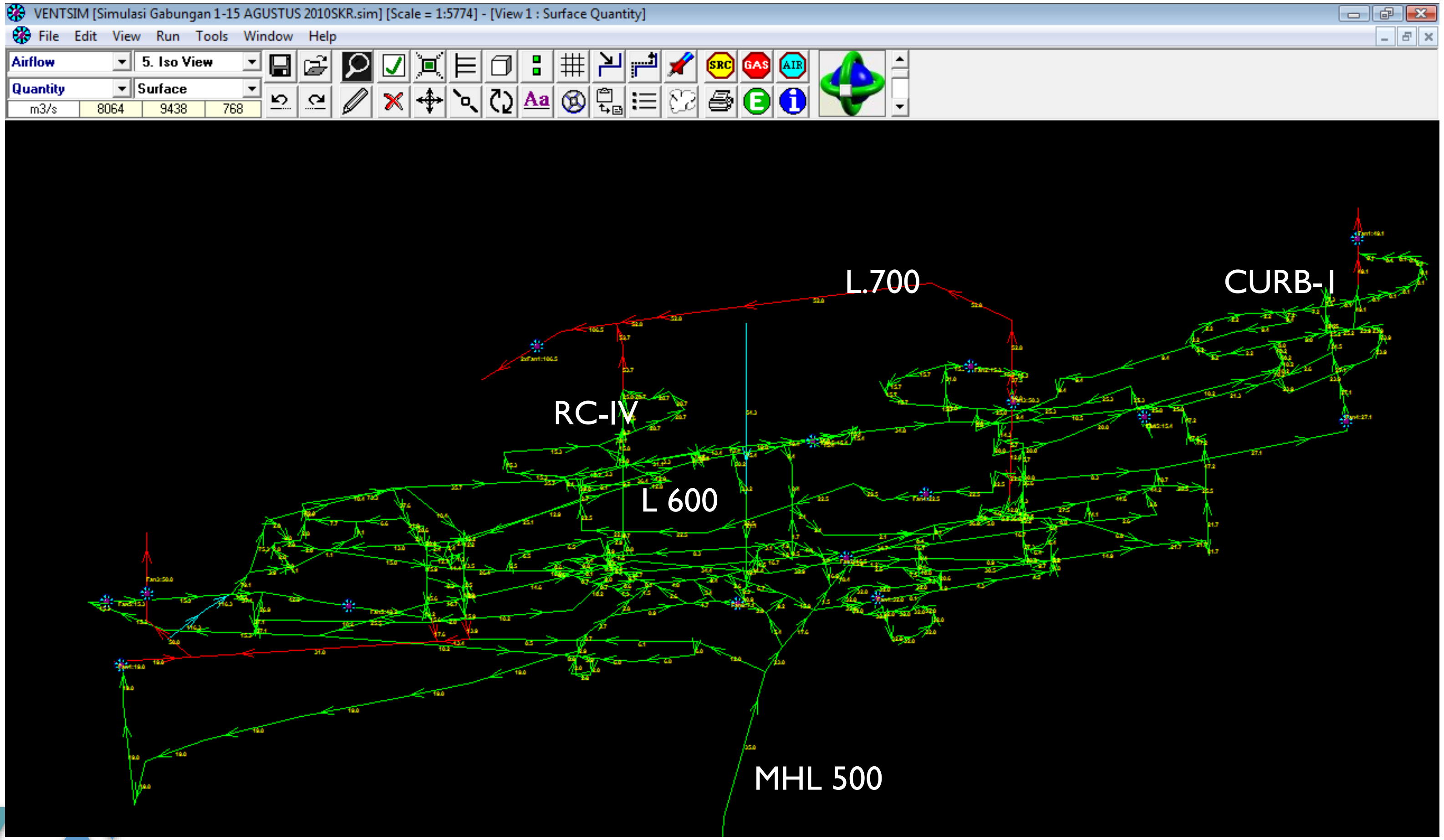
Source: Course literature

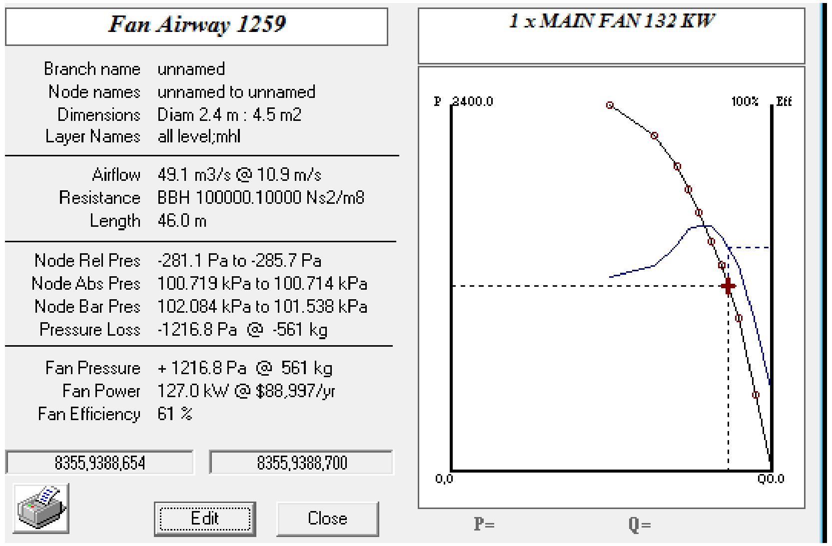
MODEL DASAR VENTILASI



Source: Course literature

SIMULASI JARINGAN VENTILASI





Source: Course literature

JALUR UDARA MASUK (INTAKE AIRWAY)

(i) Portal L. 600 (± 116,3 m^3/s)
3,3 m x 3 m pada portal dan 4 m x 3,5 m pada jarak ± 10 m setelah portal

(ii) MHL 500 (± 35 m^3/s)
Dimensi jalur ini adalah 3,3 m x 4 m

(iii) CURB-1 (± 54,3 m^3/s); diameter 2,5 m

Source: Course literature

KIPAS ANGIN UTAMA (MAIN FAN) DI CIURUG

(i) *Fan* di RC-IV L 600 (Selatan)
110 kW; 200-2400 Pa; 25-51 m^3/s

(ii) Dua buah *fan* dirangkaikan Paralel
di L 700 (Selatan), masing-masing:132 kW; 250-2500 Pa; 28-55 m^3/s

(iii) *Fan* di RC-12 (Utara) 132 kW;
200-2400 Pa; 28-56 m^3/s

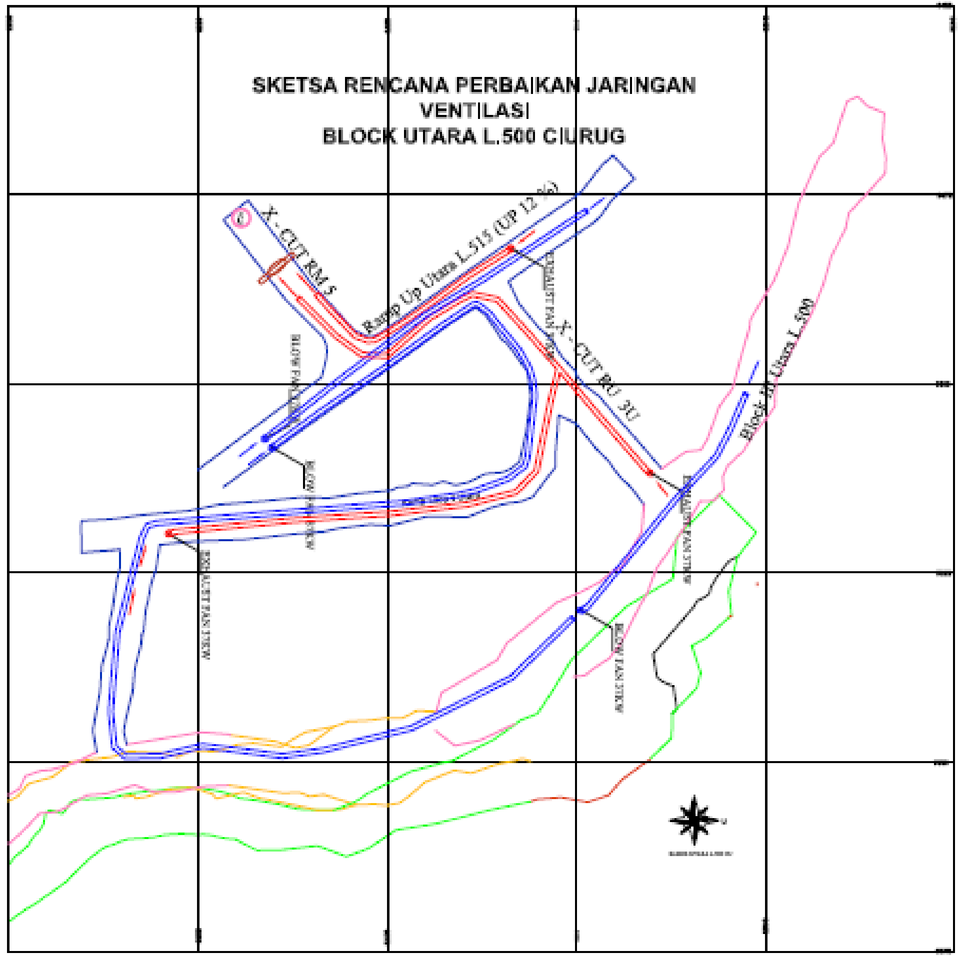
Source: Course literature

KIPAS ANGIN TAMBAHAN DAN KIPAS ANGIN PENGUAT

Kipas angin tambahan (auxiliary fan)

Pipa udara (duct) pada ventilasi lokal

Kipas angin penguat (booster fan) dan *bulkhead (*Hoffman Manufacturing, Inc. (2020)



Source: Course literature

12th U.S./North American Mine Ventilation Symposium 2008 – Wallace (ed)
ISBN 978-0-615-20009-5

Ventilation design for the Big Gossan open stope mine

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ABSTRACT: The Big Gossan mine is part of PT Freeport Indonesia's continued expansion. The deposit that will be exploited is copper-gold and will be mined using mechanized open stopping mine method. This mine will help support PT Freeport Indonesia future production together with the DOZ mine expansion and the future Grasberg Block Cave Mine, following the Grasberg Open Pit closure in about 2014. This paper presents ventilation planning for the Big Gossan Mine production and development until 2012. The mine has been in development since 2006. Production from stopes are initiated in 2009, peak production is achieved in 2011 with 7,000 tpd production rate, with the mine projected to close in 2028. This mine consists of 26 production and service levels along almost 1 km foot wall drift on each level. The present reserves for this mine exceed 53 million tonnes. The paper discusses airflow distributions to all levels necessary to provide proper ventilation to all those levels based on activity. The main fans that are used for this design are two 1600 kW mixed flow fans. Discussion is on the ventilation design criteria used, the ventilation network modelling, the proposed infrastructure requirements, ventilation control devices and recommendations for future study.

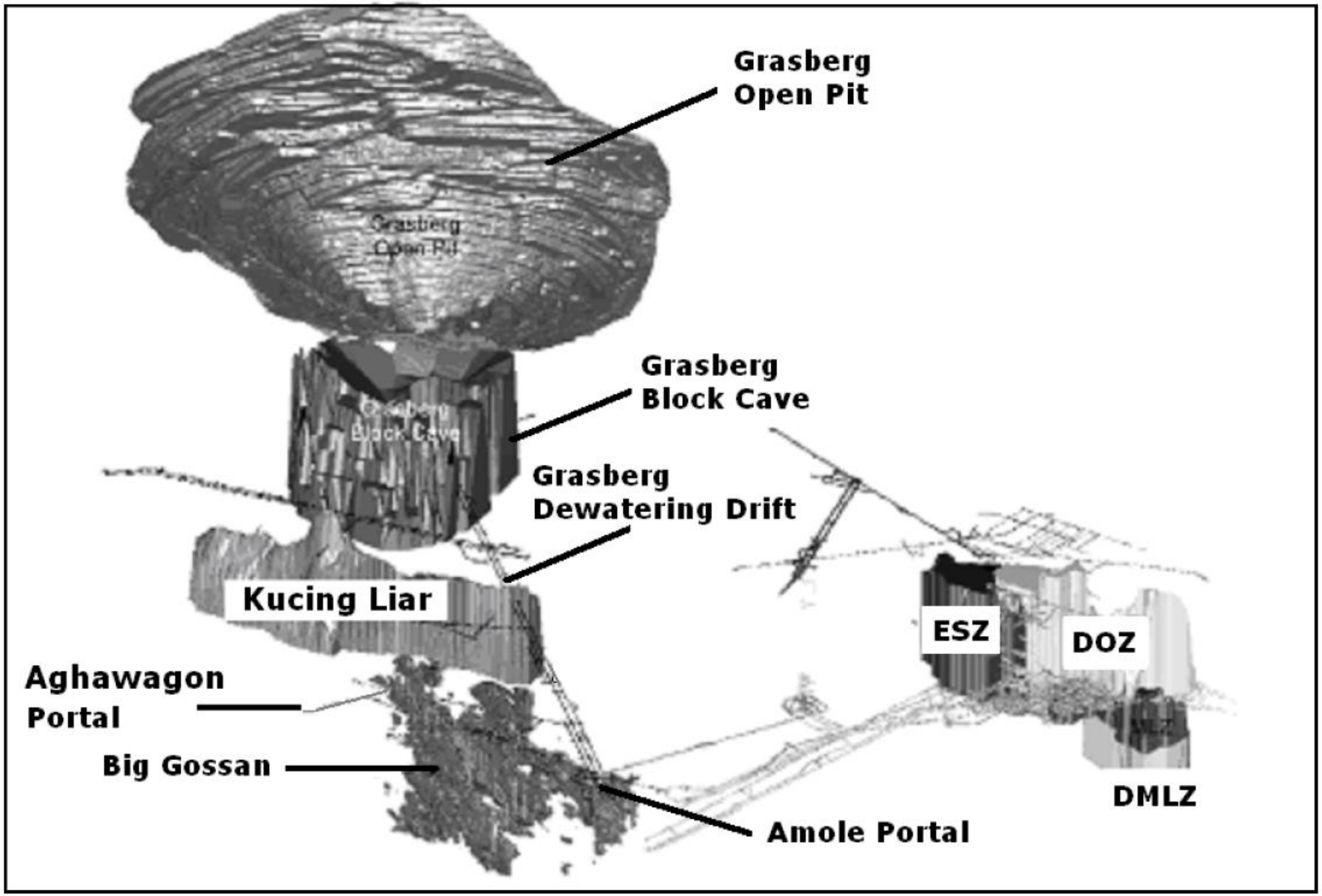

Figure 3. PTFI Ore bodies, including the Big Gossan Deposit.

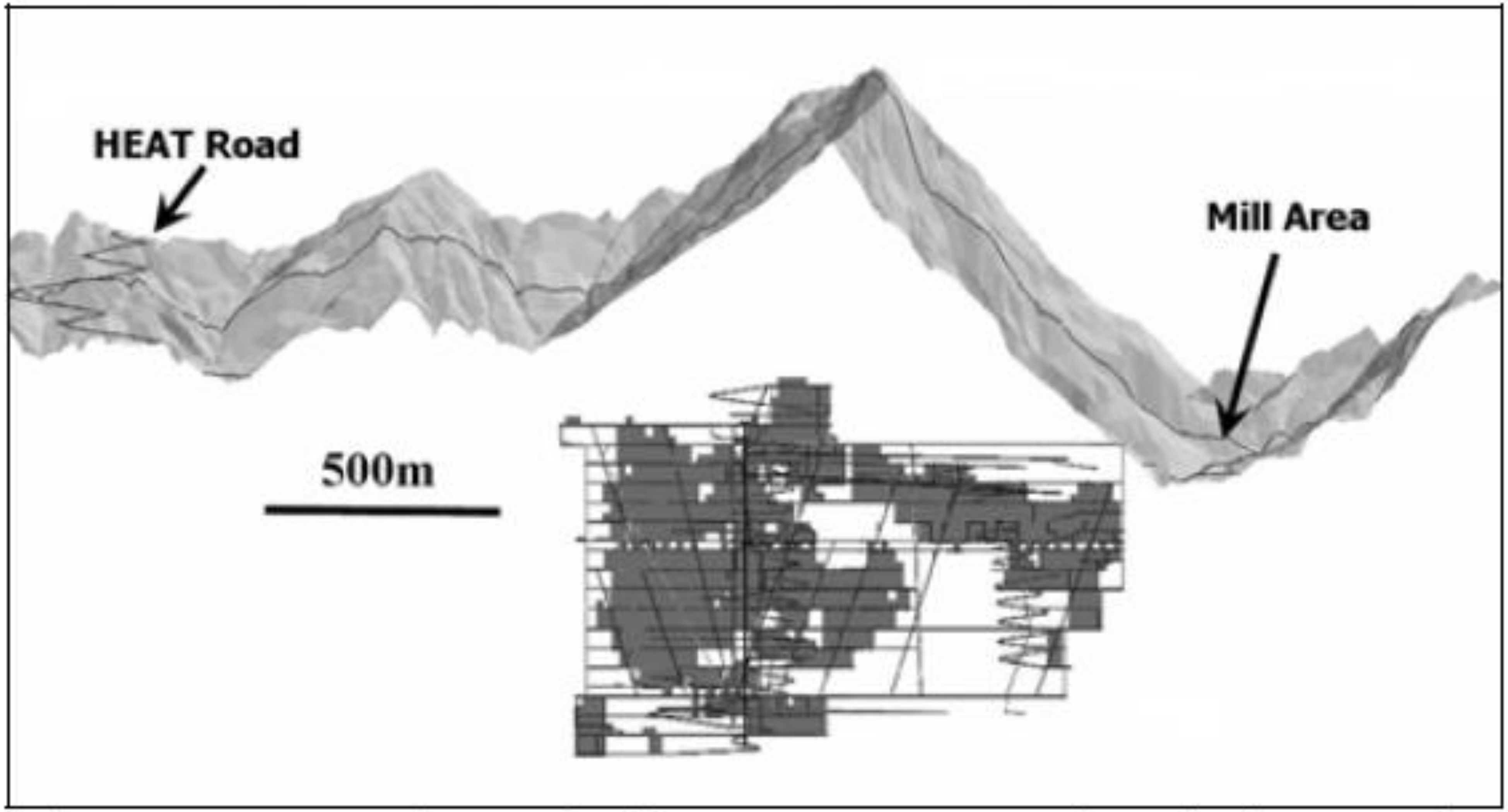

Figure 2: Location of Big Gossan Deposit relative to the Mill and surface features.

2 Big Gossan Open Stope Mine

Development of the Big Gossan Mine Began in 2006, with first production expected to occur in 2009. The peak production target of 7,000 tpd is expected in 2011 which corresponds to incremental metal of approximately 61,236 tonnes of copper and 1,843 kg of gold per year.

The Big Gossan Mine operational infrastructure includes:

- 24 different levels from 2420/L to 3180/L, mine Production and infrastructure level
- 2 ramps accessing to related levels, i.e. East & West Ramps, which also serve as secondary intake for all levels
- 9 Ore passes connecting to all Production levels, linked to Truck Haulage Level
- 4 Return Air Raises connecting to all levels
- One Fresh Air Raise connecting to all levels near to the West ramp
- An underground Crusher at 2520/L, below the Truck Haulage Level with one proposed ore bin
- Production Shaft with 2 skips, linking the Ore Bin to Conveyor Drift 2960/L
- A conveyor belt system linking the top of the production shaft to the surface milling facility
- An underground Paste Backfill Plant at 3120/L, producing the backfill material for all mined stopes

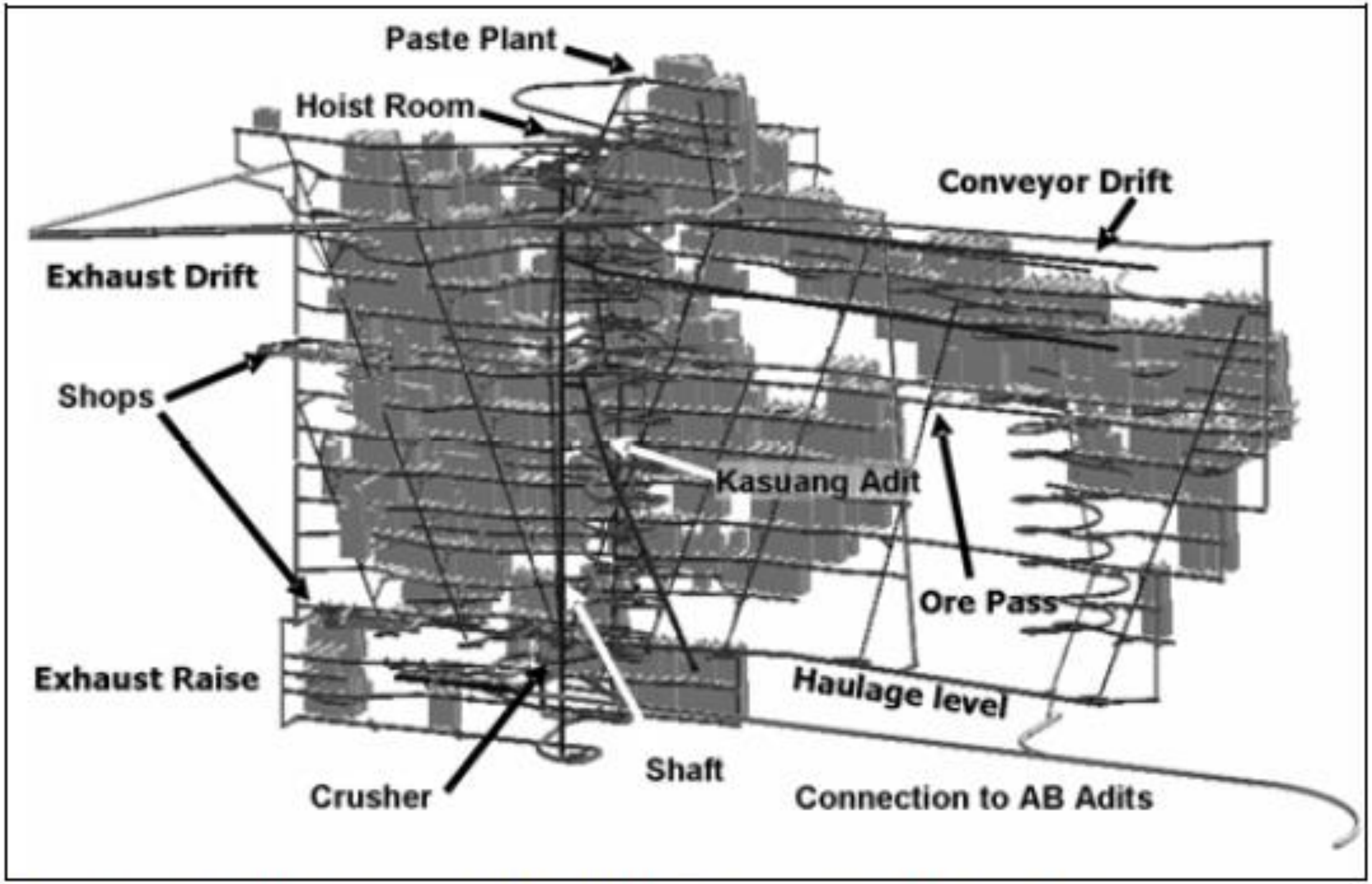

Figure 4. 3-D view of the Big Gossan Mine design.

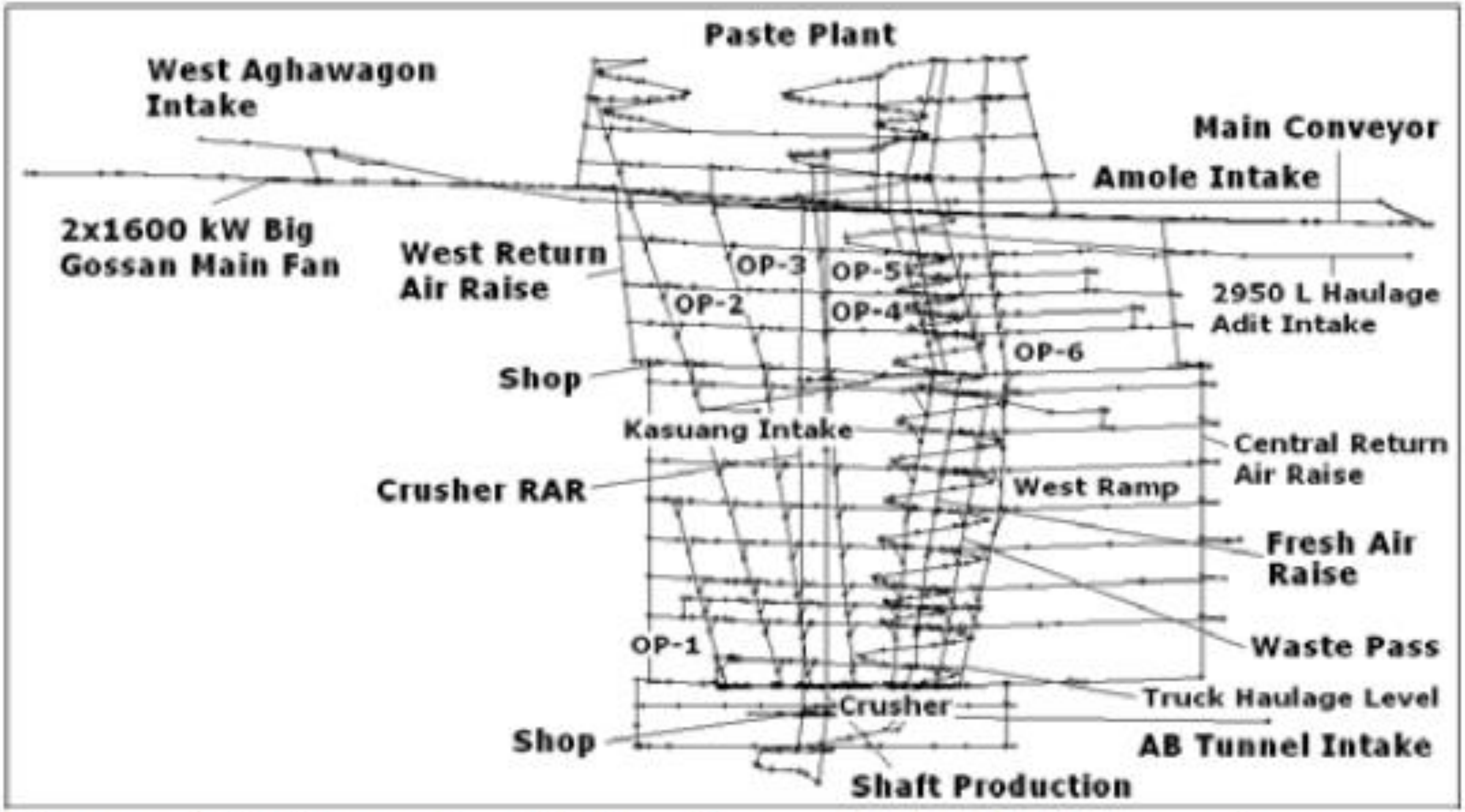

Figure 5. Schematic of Big Gossan ventilation system.

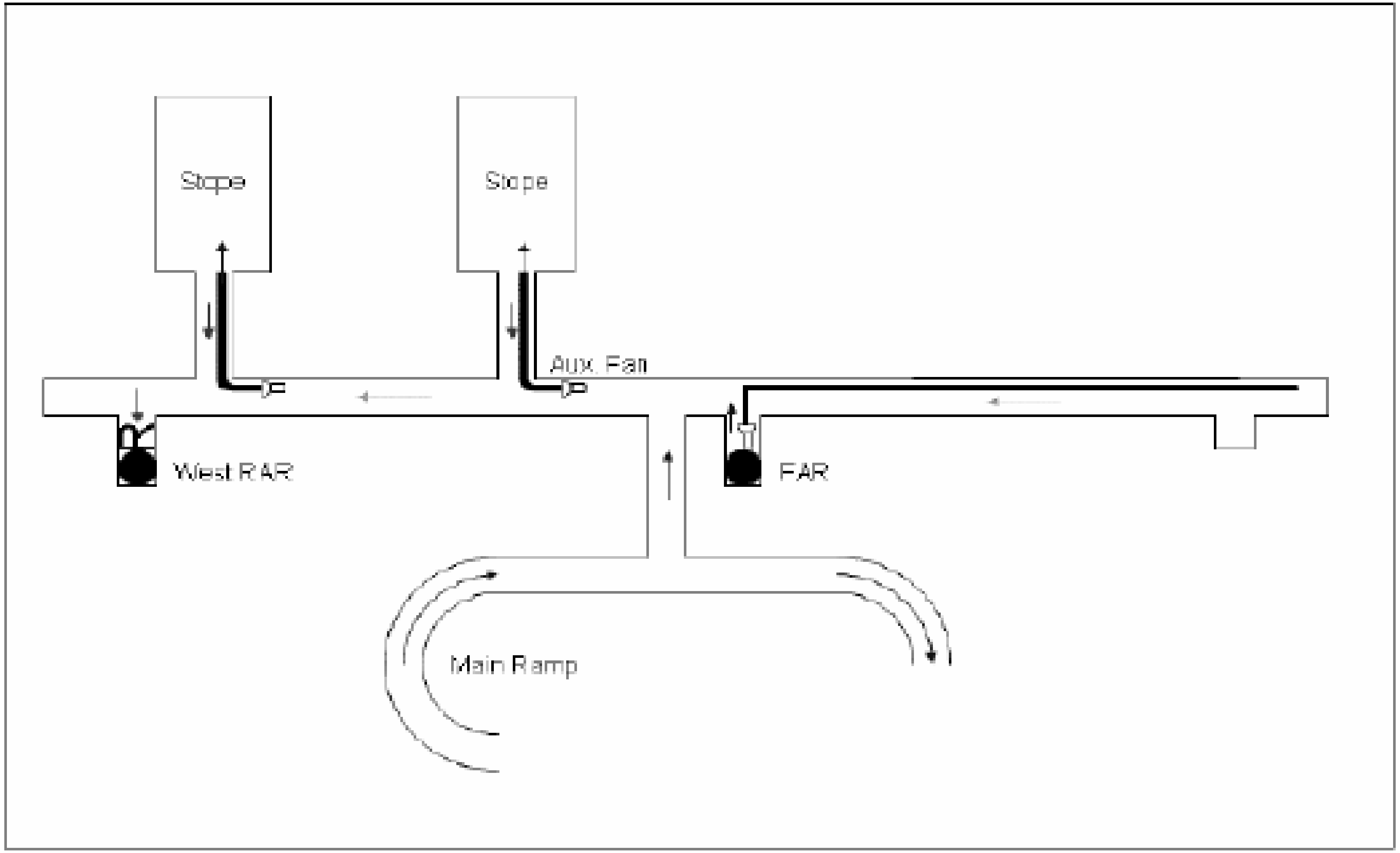

Figure 6. Typical auxiliary ventilation on Production levels.

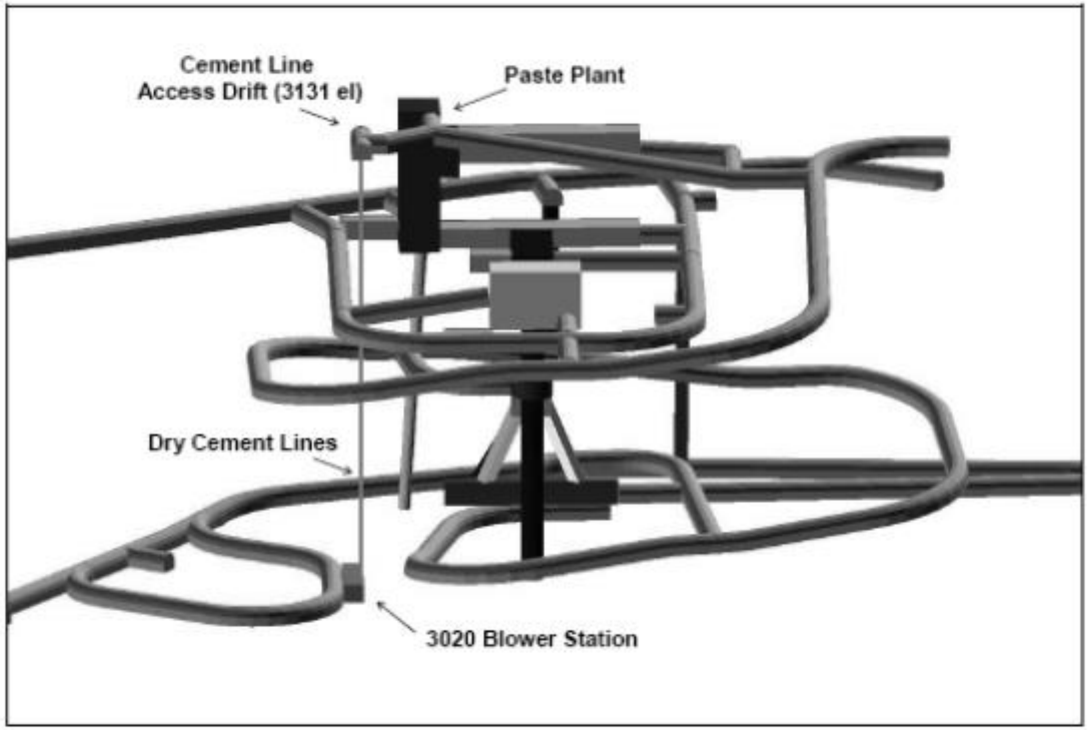

Figure 7. Isometric view of the paste plant infrastructure.

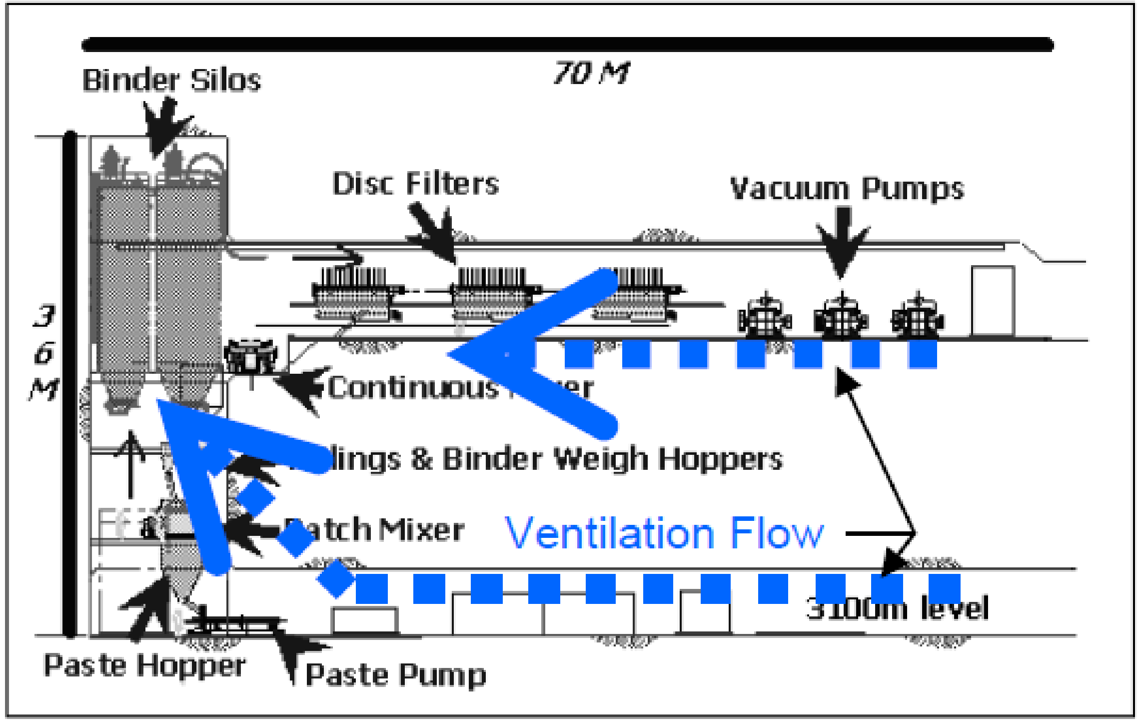

Figure 8. Paste Plant ventilation scheme.

Table 2. PTFI air velocity design criteria.

Airway	Air Velocity (m/s)		
	Min	Opt	Max
Conveyor Drifts - Homotropal - Antitropal	 0.8 0.8	 2.0 1.0	 4.0 2.0
Truck Haulage Drifts	0.8	4.1	6.1
Primary Ventilation Drifts	0.8	8.1	10.2
Rough-Walled Large Vent Raises (+4 m)	0.8	14.2	19.8
Typical ALIMAK Ventilation Raise	0.8	12.7	19.8
Drop Ventilation Raise	0.8	6.6	19.8

Table 3. Regional Atkinson friction factors.

Description	Friction Factor (kg/m^3)	
	Actual*	Standard
$9\ m^2$ to $15\ m^2$ - Drifting $15\ m^2$ to $20\ m^2$ - Drifting $20\ m^2$ to $30\ m^2$ - Drifting $30\ m^2$ and Up - Drifting	0.0102 0.0093 0.0083 0.0074	0.0132 0.0120 0.0107 0.0095
Conveyor Drift (c/w 2.1 m Belt)	0.0093	0.0120
Primary Ventilation Raise – 6.0 m ϕ+ Typical ALIMAK Raise – 3.0 m ϕ+ Drop Ventilation Raise+	0.0074 0.0111 0.0148	0.0095 0.0143 0.0191

*At average mine air density = $0.93\ kg/m^3$

+Does not include shaft/raise entry and exit losses

Table 4. Other local ventilation design criteria.

Criterion	Value
Minimum Airflow Provision per 100 Workers	7.1 m^3/s
Design Airflow per 100 kW of Diesel Equipment	7.9 m^3/s
Shops and Fixed Facilities – Direct to Exhaust	
Diesel Equipment Shop	40.1 m^3/s
Lube/Fueling Shop	28.3 m^3/s
Paste Plant	40.1 m^3/s
Explosives Magazine	9.4 m^3/s
Common Gas PELs	
Oxygen	> 19.5%
Carbon Dioxide	TWA = 5,000 ppm
Carbon Monoxide	TWA = 50 ppm STEL = 400 ppm
Hydrogen Sulfide	TWA = 10 ppm
Nitrogen Dioxide	STEL = 5 ppm

Table 5. Big Gossan Mine estimated airflow requirements for 7,000 tpd production.

Item	∑	Opt. Fact. (%)	Unit Pow. (kW)	Airflow per Unit (m^3/s)	Total Airflow (m^3/s)
Mobile Equipment					
LHD 2.7 m^3 (Clean Up) - Dsl	2	30	123	9.7	5.8
LHD 6.0 m^3 (Dev) - Dsl	5	60	201	15.9	47.8
LHD 8.2 m^3 (Prod) - Dsl	3	70	269	21.3	44.7
Drill Jumbo (Dev)	5	70	111	8.8	30.8
Production Drills	2	30	111	8.8	5.3
Drill Jumbo (Rockbolting)	2	30	111	8.8	5.3
Drill Jumbo (Sec. Breaking)	2	30	37	2.9	1.8
Truck 30 ton (Dev) - Dsl	4	60	298	23.6	56.6
Truck 30 ton (Cmnt.) - Dsl	1	60	298	23.6	14.2
Truck 55 ton (Dev) - Dsl	6	60	485	38.3	138.1
U/G Road Grader	1	40	93	7.4	2.9
Scissors Truck	1	50	130	10.3	5.1
Explsv. Truck – Dev. & Prod.	2	50	130	10.3	10.3
U/G Fork Lift	1	30	82	6.5	1.9
Shotcrete Truck - 5 m 3	3	30	120	9.5	8.5
Shotcrete Boom Truck	2	30	82	6.5	3.9
U/G Lube Truck	1	60	130	10.3	6.2
U/G Fuel Truck	3	60	130	10.3	18.5
U/G Boom Truck	1	30	130	10.3	3.1
Backhoe	1	30	82	6.5	1.9
U/G Service Truck	1	20	130	10.3	2.1
U/G Electrician Truck	1	30	130	10.3	3.1
U/G Fire Truck	1	30	130	10.3	3.1
Shop Fork Lift	2	30	32	2.5	1.5
Telehandler	4	30	78	6.2	7.4
Personnel Carrier - 30 person	4	30	150	11.9	14.2
U/G Tractor (General)	7	30	24	1.9	4.0
Toilet Service Truck	2	30	43	3.4	2.1
Personnel Vehicles	8	30	43	3.4	8.2
Subtotal Mobile Equipment					**458.3**

Item	∑	Opt. Fact. (%)	Unit Pow. (kW)	Airflow per Unit (m^3/s)	**Total Airflow (m^3/s)**
Fixed Allowances					
Paste Backfill Plant	1	100	-	40.1	40.1
Personnel	650	100	-	0.1	46.0
Maintenance Shop	2	100	-	40.1	80.2
Crshr. Bins and Conv. Drifts	2	100	-	94.4	188.8
Subtotal Fixed Allowances					**355.2**
Total Airflow Required					**813.5**

Table 6: Predicted fan duties for 7,000 tpd operation.

Fan	Pressure (Pa)	Fan Quantity (m^3/s)	Electric Power (kW)	Operating Cost (k$/yr)
Main Fan #1	1,810	460	1,250	821
Main Fan #2	1,810	460	1,250	821

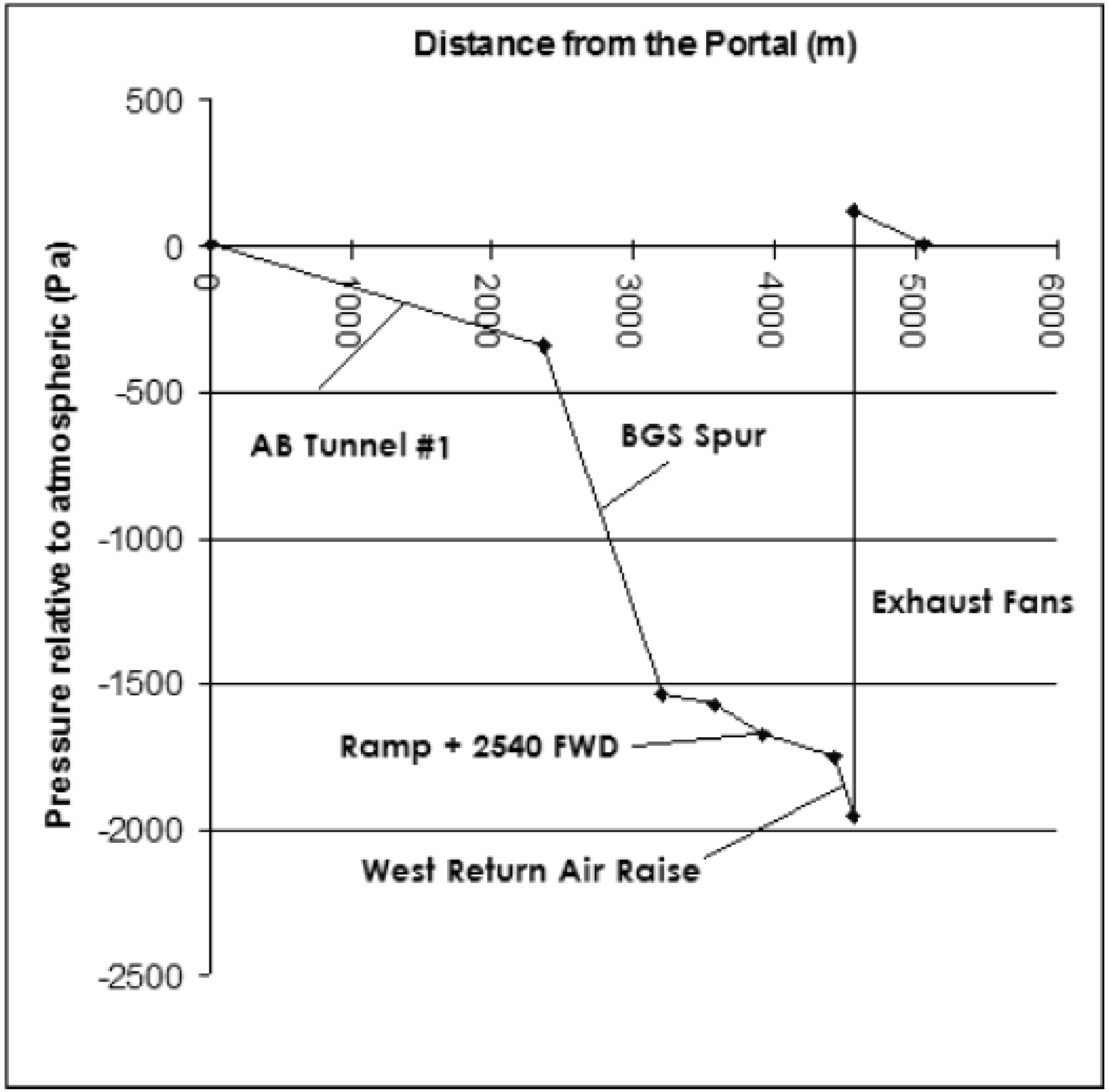

Figure 9. Pressure gradient from AB Tunnel to Main Exhaust.

The basic infrastructure sizes and design requirement for this model are:

- Dedicated Exhaust Drift, where the main fans are installed, 6.5 m x 5.5 m
- Foot Wall Drift, 5.0 m x 5.0 m
- Fresh Air Raise, 4.8 m
- Return Air Raise, 6.8 m diameter, all each connection should be equipped with a regulator
- Shaft Production, 6.0 m diameter, this shaft does not have a significant ventilation function.
- Orepasses, 3.0 m diameter, all connections to the levels should be sealed up properly, a special cover equipped with electric crane is designed to eliminate leakage through these connections
- Waste Pass, 3.0 m diameter, all these connections should be closed properly using a similar arrangement to that planned for the orepasses
- Crusher Return Air Raise, 3.1 m diameter, this raise is the exhaust raise for conveyor #522, the crusher and the explosives magazine
- Ramp, 5.0 m x 5.0 m
- Conveyor Drifts, 5.0 m x 5.0 m

3.1 A Real Time Airflow Monitoring & Control System

To reach 7,000 tpd productions there will be 9-10 active stopes. All production levels will be ventilated by approximately 23 m^3/s, and all other levels will be ventilated with 7 m^3/s. Effective flow control may be by means of regulators in all levels that are controlled based on the production schedule.

A real-time airflow monitoring and control system is highly recommended for implementation. Such a system will make distribution control easier than if it were done manually on the numerous different levels. A robust, motorized damper-type regulator would be a good choice for this purpose. Integrated control of the individual regulators should be considered to avoid damage due to the

3.2 Leakage Prevention

Based on the ventilation model, it is observed that there is a significant problem in the system associated with high levels of leakage, which is expected due the numerous connections from the foot wall lateral drifts to the vertical raises, ore passes and through the stopes. Such challenges are noted to be common with mines of this type. Some possibilities of leakage sources and the prevention/control measures include:

- Finger raises on the ore passes. These connections should be covered with steel plate that is opened during ore dumping.
- Interconnection between two levels in a production stope. The leakage control/prevention measure should include installing a full curtain at the drilling level, as a temporary bulkhead.
- Before backfill process, the open stope may be a short circuit to the level above, again the use of a curtain to control inter-level flow may be required.

3.3 Dust & Diesel Particulates

Control of dust in open stope mines is a big challenge. Dust sources expected are from loaders, trucks, crushers, belts, paste backfill plant and orepasses. In certain areas, such as belt transfers, orepass head chutes, shaft production skip dump, paste backfill plant and stockpiles engineered dust control measures may be implemented. Dust Exposure Levels should be continuously monitored, by both personal and area sampling methodologies. In the paste plant ventilation system a dust collector system is designed to remove dust exposed in the cement transfer system and also in tailing drying process. Orepasses may also contribute as dust producers by dust blow back while dumping ore at upper levels. The mining operation also strives to meet compliance with the MSHA diesel particulate exposure limits, and as such the fuels, equipment selection, operation and maintenance, and the ventilation system will factor this in.

3.4 Mixed-Flow Mine Fans

Two large-diameter mixed-flow fans are planned for the Big Gossan Mine. One of the fans has been erected and was commissioned in November 2007. The other main fan

is planned for 2010. The fans selected for the Big Gossan Mine are mechanically identical to the two 3.5m diameter mixed-flow Howden fans have already been constructed as part of the DOZ Mine expansion. During the past year these fans have proved to be capable of meeting the airflow and pressure requirement while offering the durability required by PTFI (Duckworth et al. 2006).

An example of this type of fan, under-construction, is illustrated in Figure 10. Note the impeller within the housing and the partial Radial Vane Controller located at the foreground.

Figure 10. Mixed-flow fan under-construction, with impeller and Radial Vane Control evident.

MINE VENTILATION AND AIR CONDITIONING

THIRD EDITION

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CHAPTER 14

Metal Mine Ventilation Systems

Air Quantities

In most cases, air quantities are not determined by statute for metal mines, the exception being mines where radioactivity is present. Air quantities where diesels are used are determined from manufacturers' specifications or by testing and certification done on individual units by the Mine Safety and Health Administration (MSHA). It is generally satisfactory to provide 150 cfm/bhp (1.0 $m^3/s \cdot kW$) for each unit (Kenzy and Ramani, 1980). For several units on the same split of air, full quantity requirement for the first unit, 75% of full quantity for the second unit, and 50% of the full quantity for each additional unit are recommended. Otherwise, air quantities are usually determined by practical considerations and experience.

Velocities, wherever possible, should not exceed 1500 fpm (7.6 m/s), both for comfort and for reducing head losses due to friction. This may not always be possible, particularly where a large quantity of air is needed to service an area. Air velocities at a minimum should be kept above 50 fpm (0.25 m/s), as this is about the lowest velocity at which air movement can be felt; however, this requirement may be relaxed in large service bays. In workplaces, air velocities should be 200–400 fpm (1.0–2.0 m/s) in stopes and 400–600 fpm (2.0–3.0 m/s) in drifts. These velocities are recommended for warm mines and may have to be reduced in cold mines.

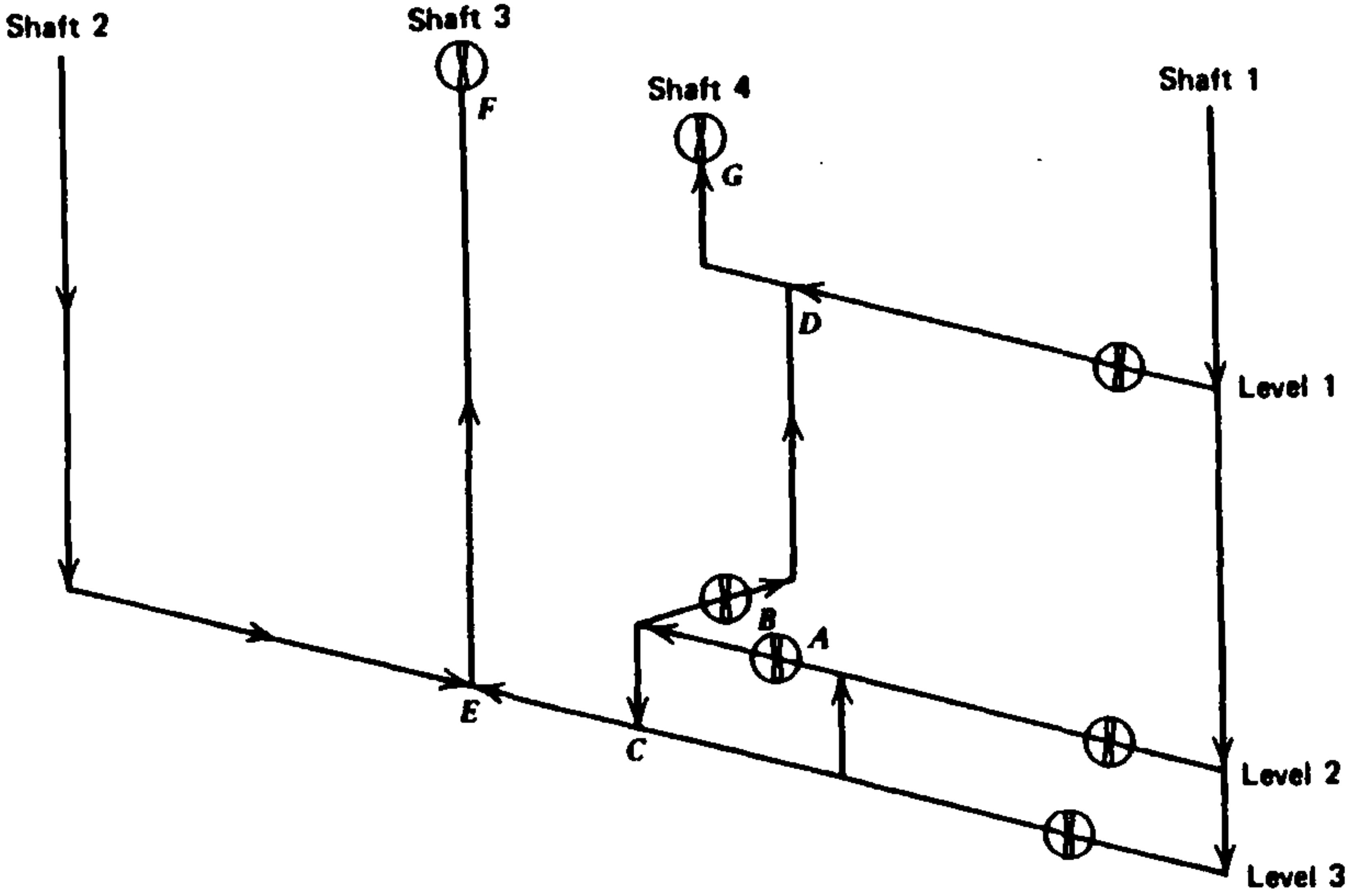

FIGURE 14.1 Ventilation schematic for a typical mature stope mine.

Leakage

As in coal mines, in metal mines leakage is the most common cause of inefficient distribution of air in mines. While leakage in coal mine ventilation systems can reach 80% of the total volume of air circulated, in metal mines, it averages about only 25%. However, in metal mines, leakages not in excess of 15% are attainable with good ventilation practice.

Threshhold Limit Values

MSHA regulations governing safety and health standards in underground metal and nonmetal mines are contained in Part 57 (CFR 30), subparts D and G (Anon., 1995a). Since the products mined in metal and nonmetal mines vary widely in terms of their mineralogy, chemical and physical characteristics, and health effects, there are not as many generalized standards (such as those for underground coal mines) applicable to all metal and nonmetal mines. For example, few air-quantity requirements are specified. However, air-quality standards do exist. For example, the minimum oxygen content must be 19.5%. With regard to exposure limits on air contaminants, MSHA must be 19.5%. With regard to exposure limits on air contaminants, MSHA has adopted the threshold limit values (TLVs) set forth by the American Conference of Governmental Industrial Hygienists (Section 3.2). The TLV-TWA for radon is 1.0 working-level maximum. In 1989, MSHA issued a proposed rule for air quality, chemical substances, and respiratory protection standards (Anon., 1989b). These standards are under consideration for final adoption. Two factors unique to deep mines are heat and humidity, and these must be addressed adequately for both health and productivity reasons (Johnson, 1992b). As with any planning exercise, information on the most recent ventilation and air-quality standards and new, more effective industry practices must be collected.

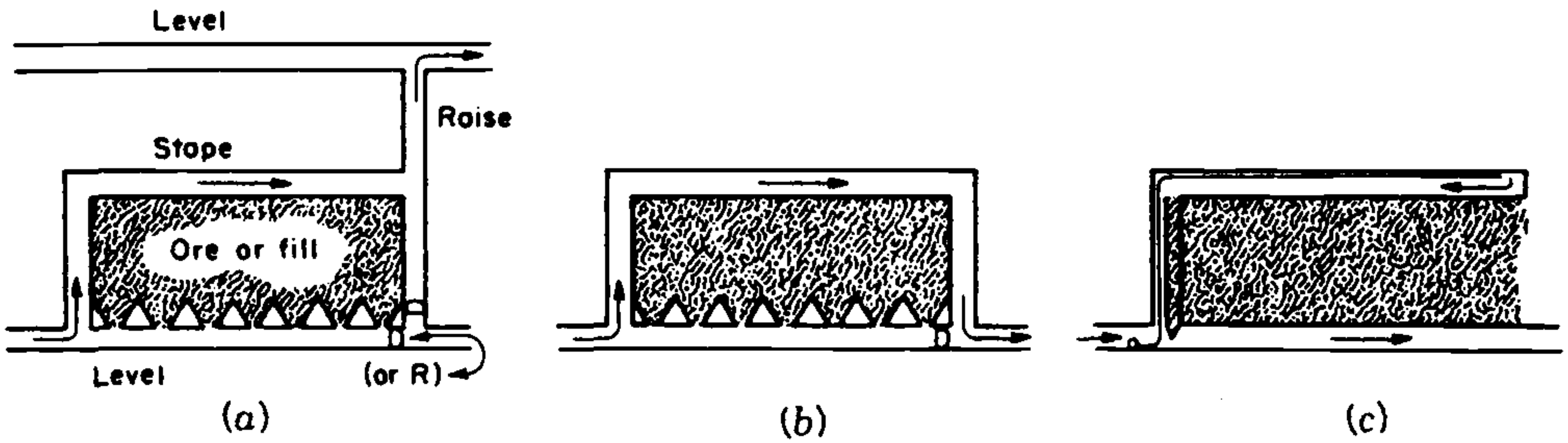

FIGURE 14.2 Ventilation in workplaces for mining methods in a near-vertical plane: (*a*) stopes with interlevel connections; (*b*) stopes with no connections but two access openings; (*c*) stopes with no connections but one access opening.

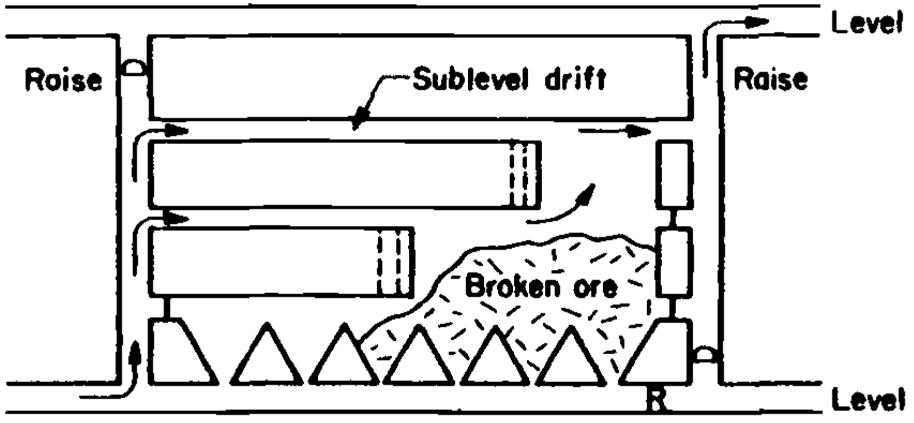

FIGURE 14.3 Ventilation system for sublevel stoping.

Block Caving Mines

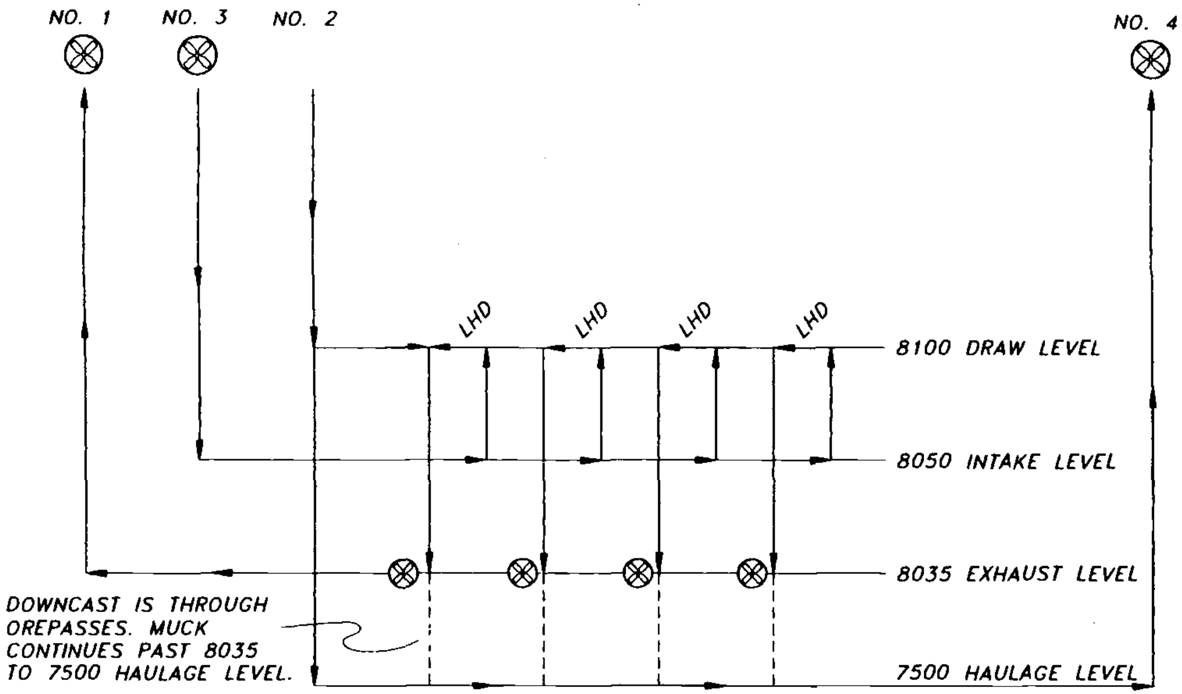

FIGURE 14.9 Ventilation layout of Henderson mine (Knape, 1985).

Henderson Mine Air is downcast through No. 3 shaft and directed across the 8050 intake level to the production area (Fig. 14.9). Air is then upcast to individual draw drifts on the 8100 level, where LHDs are in use. Fresh air travels past the LHD and downcasts the orepass in which the LHD is dumping and is drawn off the orepass on the 8035 exhaust level by booster fans; the muck continues down the orepass to the 7500 haulage level. Air travels across the 8035 level to exhaust at No. 1 shaft. Main intake fans are located at the collar of No. 3 shaft and main exhaust fans at the collar of No. 1.

The 7500 haulage level is ventilated by a separate intake shaft, No. 2, and a separate exhaust shaft, No. 4. Motivation for this system is by exhaust fans at the collar of No. 4.

14.7 DESIGN EXAMPLE

A steeply dipping copper ore body Fig. 14.16 is to be mined by the overhand cut and fill method using LHD machinery in the stopes. Each stope is to be 200 ft (61 m) long, developed in pairs, with a 3-ft (0.9-m) round airway on the ends of each stope pair, and a 3-ft (0.9-m) orepass–ventilation borehole in the center. A 100-ft (30-m) pillar is to be left between each stope pair. Main-level haulage will be by battery-powered vehicles. Production is designed for 14 active stopes.

Drifts are timbered and driven 10 by 10 ft (3.0 × 3.0 m). The intake-hoisting shaft is 20 ft (6.1 m) in diameter and concrete-lined. Similar shafts of this type have been found to have a resistance $R = 0.01 \times 10^{-10}$ in·min^2/ft^6 per 100 ft (3.665×10^{-3} N·s^2/m^8 per 100 m). The exhaust shaft is to be round, concrete-lined, 8 ft (2.4 m) in diameter, and with no furnishings except a small cage guide for emergency service and shaft inspection.

Design a ventilation system and specify regulators and fans suitable for this mine. Assume that shaft collars are both at sea level, and ignore air specific-weight changes, shock losses, and velocity heads.

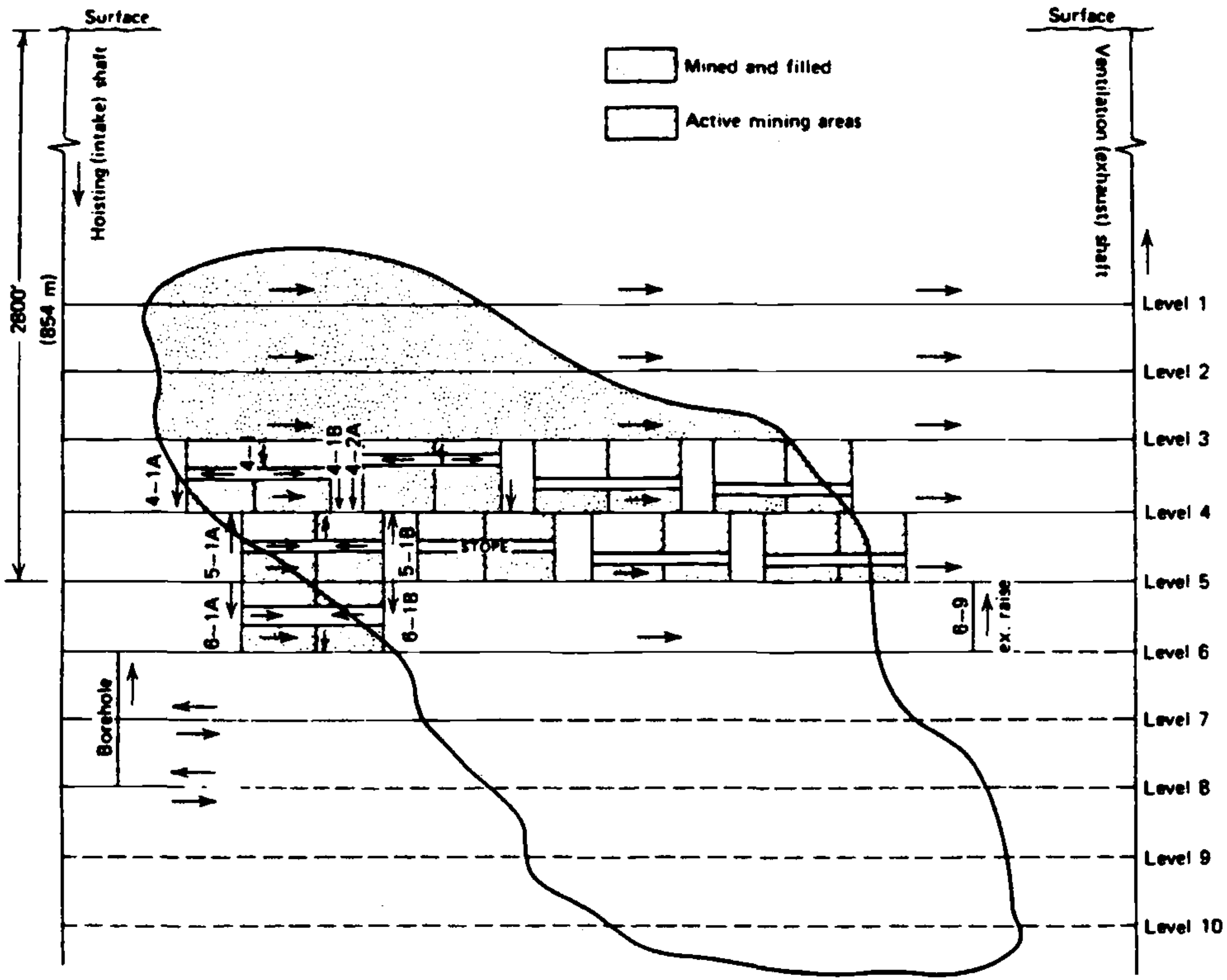

FIGURE 14.16 Simplified ventilation scheme for cut and fill mine (see Example 14.1).

Solution

A period about halfway through the life of the mine will be used as the point at which this problem will be solved, shown in Fig. 14.16. The ore has been mined out between levels 1 and 3; levels 3–5 are in full production, and level 6 is being driven toward the exhaust shaft past the last exhaust borehole, which has just been cut. Stope pair 6A–6B is being started. Levels 7 and 8 are being driven, and levels 9 and 10 have not been started. Levels 1 and 2, although no longer active, are still maintained.

A review of Example 6.7 would be helpful before proceeding. Since diesel exhaust is contaminating the air, none of the air that has passed through the stopes will be reused. A ventilation system using alternate levels as intakes with the intermediate levels as exhausts is selected. Levels 3 and 5 are intakes, and levels 4 and 6 exhausts. A small amount of air to ventilate the exhaust levels from the shaft to the stoping areas is necessary as well.

Step 1. Lay out the Ventilation Pattern Level 5 will upcast through the stopes via the supply raise at the end of each stope pair and exhaust through the center boreholes. Level 3 will downcast through the center borehole and exhaust through the supply raises down to level 4. These are shown in Fig. 14.17.

Step 2. Calculate Needed Air Quantities The required air quantities in the workplaces are determined by application of various criteria. One such criterion, for example, would be for the diesel LHD equipment. By consulting USBM Schedule 24, it is found that Approval No. 24-187 covers these specific units and calls for a minimum of 5600 cfm per unit (Anon., 1980). For an additional margin of safety, use 7000 cfm per unit, or 14,000 cfm per stope pair. Other design criteria are a minimum of 100 fpm air velocity in fresh-air drifts and 200 fpm in exhaust drifts.

The following table shows how individual air-quantity requirements were selected for the mine. The magnitude and direction of these flows have been plotted on Fig. 14.17.

Workplace	Controlling Criterion for Velocity	Air Quantity, cfm
Level 1	100 fpm minimum	10,000
Level 2	100 fpm minimum	10,000
Level 3	Diesel equipment in stopes—200 fpm minimum	76,000
Level 4	200 fpm minimum	20,000
Level 5	Diesel equipment in stopes—200 fpm minimum	90,000
Level 6	200 fpm minimum	10,000[a]
Level 7	100 fpm minimum	10,000
Level 8	100 fpm minimum	10,000
		Total 236,000
		[Will round off to 250,000 cfm (118 m^3/s)]

[a] Requirement on level 6 is 20,000 cfm, which can be entirely supplied from levels 7 and 8. Nonetheless, 10,000 cfm is provided as an additional safety factor.

Step 3. Calculate Airway Head Losses and Fan Heads In many actual metal mine ventilation systems, the number of interconnecting and overlapping circuits makes the determination of the mine static head a difficult problem. For example, 28 different airflow paths exist between the intake and exhaust shafts in this problem.

The task is to find the free split, and in this case, this is best accomplished by eliminating several possible splits by visual examination of the data. High airflow quantities are noted on levels 4 and 5, so that the highest-pressure

drop (or head loss) branch is probably located in this area. Therefore, calculate head losses along these levels in the shaft and through the stopes.

(a) Find the head loss in the intake shaft, using Eq. 7.7.

$$R = 0.01 \times 10^{-10} \text{ in.·min.}^2/\text{ft}^6 \text{ per 100 ft}$$

$$H_l = RQ^2$$

Although we could calculate head losses for each segment going down the shaft, the shaft resistance between levels is so small that we can assume without introducing significant error that the entire 250,000 cfm is downcast to either level 4 or level 5:

$$H_l = (0.01)(10^{-10})(250{,}000)^2 = 1.75 \text{ in. water (level 5)}$$

Similarly,

$$H_l = 1.625 \text{ in. water (level 4)}$$

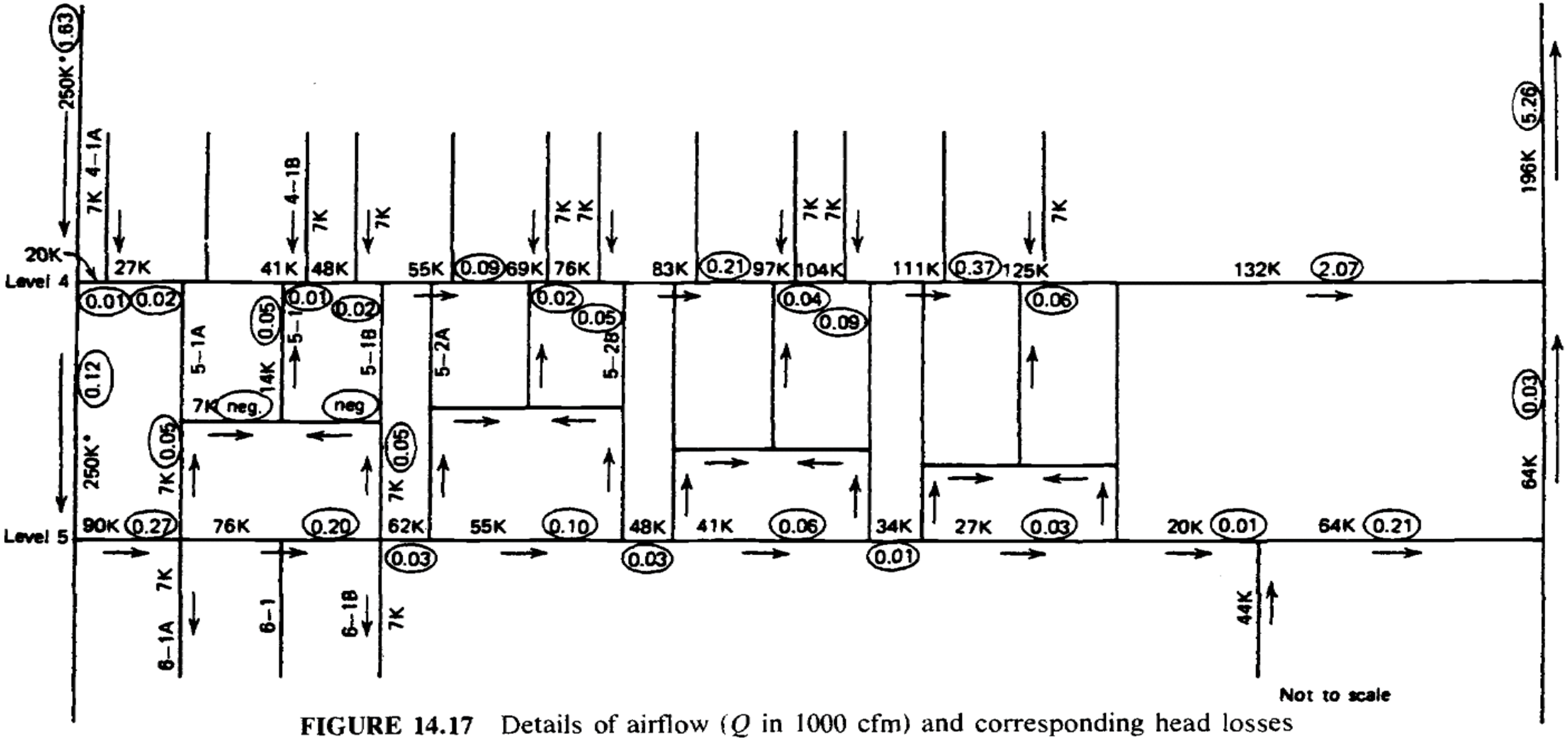

FIGURE 14.17 Details of airflow (Q in 1000 cfm) and corresponding head losses (H in in. water circled) for level 4 and 5 in cut and fill mine (see Example. 14.1). The asterisks indicate that for calculation purposes, it should be assumed that the entire 250,000 cfm (118 m^3/s) is downcast.

(b) Find the losses across levels 4 and 5, using the following K values:

$K \times 10^{10}$ lb·min²/ft⁴	Airway
110	Main levels—timbered drift
20	Access raises—bored
100	Stopes

The following head losses are calculated using Eq. 5.20:

Level 4

Airway	L ft	Q cfm	H_l in. water
Shaft to			
4-1A raise	200	20,000	0.01
4-1A to 5-1	350	27,000	0.02
5-1 to 4-1B	50	41,000	0.01
4-1B to 4-2A	100	48,000	0.02
4-2A to 5-2	350	55,000	0.09
5-2 to 4-2B	50	69,000	0.02
4-2B to 4-3A	100	76,000	0.05
4-3A to 5-3	350	83,000	0.21
5-3 to 4-3B	50	97,000	0.04
4-3B to 4-4A	100	104,000	0.09
4-4A to 5-4	350	111,000	0.37
5-4 to 4-4B	50	125,000	0.06
4-4B to exhaust shaft	1400	132,000	2.07

Level 5

Airway	L ft	Q cfm	H_l in. wate
Shaft to			
5-1A raise	400	90,000	0.27
5-1A to 5-1B	400	76,000	0.20
5-1B to 5-2A	100	62,000	0.03
5-2A to 5-2B	400	55,000	0.10
5-2B to 5-3A	100	48,000	0.02
5-3A to 5-3B	400	41,000	0.06
5-3B to 5-4A	100	34,000	0.01
5-4A to 5-4B	400	27,000	0.03
5-4B to 6-9 raise	400	20,000	0.01
6-9 raise to exhaust shaft	800	64,000	0.28

Average head losses for airflow through the stopes are as follows:

Airway	L, ft	Q, cfm	H_l, in. water
Access raise	100	7,000	0.05
Stope	100	7,000	negl.
Fill raise	100	14,000	0.05

(c) Find head losses in exhaust shaft. Assume an 8-ft diameter exhaust shaft with $K = 10 \times 10^{-10}$ lb·min²/lb⁴, yielding an R value of 0.038×10^{-10} in·min²/ft⁶ per 100 ft. Thus head losses are

Airway	L, ft	Q, cfm	H_l, in. water
Level 5–level 4	200	64,000	0.03
Level 4–level 3	200	196,000	0.29
Level 3–level 2	200	216,000	0.35
Level 2–level 1	200	226,000	0.39
Level 1–surface	2000	250,000	4.75
			5.81 in.

Some of these values are plotted on schematics of levels 4 and 5 in Fig. 14.17.

Step 4. Determine the Free Split By calculating the cumulative head loss for the various paths across levels 4 and 5, one can identify the highest head loss branch as

$$\text{Intake} \rightarrow \text{level 5} \rightarrow \text{raise 5-2B} \rightarrow \text{level 4} \rightarrow \text{exhaust shaft}$$

This path has a total head loss of

Intake shaft	1.75 in. water
Level 5	0.60
Between levels	0.10
Level 4	2.91
Exhaust shaft	5.81

$$H_l = 11.17 \text{ in. water (2.792 kPa)}$$

To obtain the prescribed airflow through every airway, only the free split (highest head-loss branch) will require no regulation.

Step 5. Specify Fans Now that the mine static head has been determined to be 11.17 in. water (say, 11 in. water), the fans must be specified. It is decided to use one booster fan in an intake position on each level and a single exhaust fan on the exhaust shaft. To maintain positive pressure on

the working areas, the booster fans could be selected to operate against 5.5 in. water, and the exhaust fan would be specified to deliver 250,000 cfm at 5.5 in. water. With this configuration, failure of the exhaust fan will still result in a fair amount of ventilation in the mine.

Because the pressure difference between intake and exhaust levels is so slight (0.1 in. water), fans producing the same pressure should be selected for each level. Thus fan requirements are as follows:

Location	Q cfm	(m^3/s)	H_s in. water	(kPa)
Level 1	10,000	(4.72)	5.5	(1.369)
Level 2	10,000	(4.72)	5.5	(1.369)
Level 3	76,000	(35.87)	5.5	(1.369)
Level 4	20,000	(9.44)	5.5	(1.369)
Level 5	90,000	(42.48)	5.5	(1.369)
Level 6	10,000	(4.72)	5.5	(1.369)
Exhaust shaft	250,000	(117.98)	5.5	(1.369)

The fans for levels 3 and 5 can probably be of the same type with different blade pitch settings. Levels 1, 2, and 6 can also use the same type. Levels 7 and 8 at this time need auxiliary fans blowing through tubing, with another auxiliary fan on level 6 blowing air to their intakes. This is an auxiliary-fan problem and is not treated here.

The fan selection procedure illustrated in Section 13.15 can be used here to select the main fan. The relative large head loss in the exhaust shaft suggests that perhaps a larger-diameter shaft could be justified. By reviewing the principles contained in Chapter 12 and current shaft-sinking or raise-boring costs, the reader can determine the most economical size for the exhaust shaft.

Tugas 4 (Perorangan)

- Kerjakan kembali contoh soal 14.7 dengan penjelasan untuk setiap tahapnya.
- Buatlah analisis dan kesimpulan dari sistem ventilasi tersebut **(tidak diperkenankan sama untuk setiap mahasiswa, karena akan menjadi bukti kerjasama).**

Keterangan:

- Gunakan satuan dalam SI.
- Tugas dikerjakan dalam bentuk word/pdf dan dilengkapi perhitungan dalam excel.
- Tugas diupload ke Google Drive dan diberikan nama files sesuai NIM setiap mahasiswa (Ketua Kelas mohon mengatur)
- Link Gdrive mohon disampaikan kepada dosen pengampu (Dr. Nuhindro P.W) oleh Ketua Kelas paling lambat tanggal **6 April 2023 pukul 17.00 WIB.**

Terima kasih